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TECHNICAL AND MARKETING
ASSESSMENT OF THE LB PROCESS

FINAL REPORT
BY

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SURVOL

Evaluation du procédé LB: aspects techniques et marketing

Les tâches de la première phase étaient les suivantes: revoir les données fournies par les expériences à échelle de laboratoire; créer un dessin proposé en vue d'une première installation commerciale et d'installations ultérieures; identifier des domaines problématiques, des limitations physiques et des considérations d'agrandissement d'échelle; et définir les chances du succès technique.

A partir de cette base, les coûts du procédé LB, avec insistance particulière sur la demande énergétique, devaient se comparer à ceux d'autres procédés existants ou en voie de développement. Un taux de rentabilité devait être préparé pour le procédé.

Le sous-traitant, Hatch, n'a pu achever l'évaluation prévue. Sur examen des données expérimentales, et suite à des discussions avec les inventeurs, un concept d'installation de première phase a été développé. La superficie requise pour le transfert de chaleur dans les tubes de carbure de silicium était si importante aux températures où les données sur le taux de réaction avaient été développées, que le nombre de tubes exigées pour produire 300 000 tonnes par an était de 80, utilisant des tubes un peu plus grands que ceux qui sont actuellement disponibles sur le marché. L'approche des tubes multiples horizontaux a été abandonnée comme étant impraticable sur le plan mécanique.

OVERVIEW

Technical and Marketing Assessment of the LB Process

The first stage tasks were to review the available data from the bench-scale experiments, to arrive at a suggested configuration for a first commercial plant and for subsequent plants, to identify problem areas, physical limitations and scale-up considerations and define the prospects for technical success.

From this base, costs for the LB Process with specific emphasis on energy demand, were to be compared to other existing and developmental processes. A return on investment was to be developed for the process.

Hatch, the subcontractor, was unable to complete the assessment as planned. Upon examination of the experimental data, and after discussions with the inventors, a first stage plant concept was developed. The surface area required for heat transfer into the silicon carbide retort tubes was so great at the temperatures where reaction rate data had been developed, that the number of retort tubes to produce 300 000 tonnes per year was 80, utilizing tubes somewhat larger than are presently, commercially available. The multiple horizontal tube approach, was abandoned as mechanically impractical.

The inventor suggested an annular vertical kiln process utilizing silicon carbide tube walls of very large diameter for the inside and outside walls with a kiln temperature ranging from 1500° C to 1700° C. A narrow annulus would serve as the reaction chamber. Preliminary experimental trials were run on static cylinders with the fine ore-coal mixture to determine reaction rates.

L'inventeur a suggéré un procédé de four annulaire vertical utilisant pour les parois intérieure et extérieure des parois de tube de carbure de silicium de très grand diamètre, et une température de four de 1500°C à 1700°C. Un mince anneau servirait de chambre de réaction. Des essais préliminaires ont été faits sur des cylindres statiques avec le mélange fin de minerais et de charbon pour déterminer les taux de réaction.

Des mécanismes et modèles possibles de transfert de chaleur ont été étudiés et discutés. Les données expérimentales sont très limitées, et absentes pour la zone de température postulée. Beaucoup de travail expérimental est exigé pour faire avancer le concept jusqu'au stade où le puits annulaire peut être considéré prêt aux essais d'installation pilote. Ainsi nous recommendons que des expériences plus approfondies soient faites pour définir les éléments suivants: taux de réaction, réactions éventuelles du produit avec le tube, caractéristiques de gonflage et d'alimentation des mélanges, taux véritables de transfert de chaleur et de réaction à travers la couche comprimée. Ces questions, et puet-être d'autres encore, devrait être résolues avant qu'une installation pilote de plus grande dimension ne soit conçue.

G. A. Chapman

GAC:NL
le 7 janvier 1988

Possible heat transfer mechanisms and models were investigated and discussed. There is very limited experimental data, and none at the postulated temperature range and much experimental work is required to pursue the concept to the stage where the annular shaft can be considered ready for pilot plant trials. Therefore, our recommendation is that more extensive experimental work be done to define reaction rates, possible reactions of the product with the retort, swelling and feeding characteristics of the mixtures and the actual heat transfer mechanism and reaction rates through the packed bed. These, and perhaps other items should be determined before a larger pilot plant is designed.

G. A. Chapman

GAC:CO
January 7, 1988

RESUME DE DIRECTION

But

La présente étude intitulée "Evaluation du procédé LB: aspects techniques et marketing" a été effectuée afin de savoir si le procédé LB pouvait se prêter à l'exploitation commerciale si le procédé était développé avec succès à l'échelle d'une installation pilote. La détermination de la praticabilité commerciale était essentielle au financement de la phase de l'installation pilote--5,9 millions de dollars (dollars 1986). Les investisseurs éventuels exigeaient des assurances dans les domaines de la technique et du marketing que, si le développement en installation pilote réussissait, le procédé serait soumis à une exploitation commerciale, ce qui justifierait l'investissement fait au développement.

Portée

La présente étude a été parrainée par Energie, mines et ressources Canada (EMR) et Stelco Inc. en vue d'examiner la praticabilité technique du procédé LB à l'échelle commerciale (300 000 tonnes par an) et de décider du potentiel en marketing qu'aurait le procédé s'il était développé avec succès. Le contrat passé entre EMR et Stelco comprenait l'emploi de Hatch Associates Ltd. de Toronto comme sous-traitant principal chargé d'effectuer une partie importante de l'étude.

Par accord mutuel l'étude a été terminée à la fin de novembre lorsqu'il a été déterminé que des recherches de laboratoire supplémentaires étaient nécessaires afin de définir avec plus de précision le procédé et ses paramètres opérationnels.

EXECUTIVE SUMMARY

Purpose

This study titled "Technical and Marketing Assessment of the LB Process" was carried out to determine if the LB Process had the potential for commercial exploitation if the process was developed to a pilot plant scale successfully. The determination of the commercial feasibility was essential to the pilot plant phase funding of $5.9 million (1986 dollars) as potential sponsors required technical and market assurance that if the pilot plant development was successful the process would be commercially developed, hence justifying the investment in the development.

Scope

This study was jointly sponsored by Energy, Mines and Resources Canada and Stelco Inc. to investigate the technical feasibility of the LB Process on a commercial scale (300 000 tonnes per year) and to determine the marketing potential for the process if it was successfully developed. The contract between EMR and Stelco included using Hatch Associates Ltd. of Toronto as the main subcontractor responsible for carrying out much of the study.

By mutual agreement the study was terminated at the end of November when it was determined that further laboratory research was required to more precisely define the process and its operating parameters.

Historique

Le procédé LB est un procédé de réduction directe et de fonte de matières ferreuses qui emploie le charbon comme source d'énergie principale pour la réduction et l'éléctricité pour la fonte afin d'élaborer un produit liquide ferreux contenant de 0.3% à 4,0% de carbone. Le procédé LB a été inventé par des chercheurs de l'Université McMaster--W. K. Lu et C. Bryk--qui faisaient des recherches sur des façons d'éliminer le besoin de fours à coke pour la fabrication future du fer. L'équipe de McMaster a construit une série d'unités LB à échelle de laboratoire afin de faire les recherches nécessaires pour définir le procédé. Au début de 1986 l'unité LB à échelle de laboratoire avait produit une quantité importante de données et la décision a été prise d'étudier des mécanismes de financer et de construire une installation pilote de 2000 tonnes par an. Stelco a fourni des données technogéniques préliminaires d'une telle installation; ces renseignements ont été incorporés à un projet soumis par Stelco à EMR sous la forme d'une demande de fonds dans le cadre du programme de Recherches et développement en matière d'énergie industrielle. L'examen de la demande a mis en relief le manque d'une analyse du marché pour appuyer l'investissement. L'EMR s'étant intéressé au projet, une étude co-financée par EMR et Stelco a été proposée afin de trouver les réponses aux questions qui s'imposaient dans les domaines de la technique et du marketing.

L'étude co-financée a été entreprise par Stelco dans le cadre du contrat du département d'Approvisionnements et services Canada (ASC) passé avec Hatch Associates Ltd. de Toronto en tant qu'élément principal de l'étude supervisée par Stelco.

Background

The LB Process is a direct reduction and smelting process for iron bearing materials using coal as the main energy source for reduction and electricity for smelting to produce a liquid ferrous product with 0.3 to 4.0 percent carbon. The LB Process was invented by W. K. Lu and C. Bryk of McMaster University who were carrying out research on how the need for coke ovens could be eliminated for future iron production. The McMaster team constructed a series of bench scale LB units to carry out the research necessary to define the process. By early 1986 the bench scale LB unit had produced a significant amount of data and it was decided to investigate mechanisms to fund and construct a 2000 tonne per year pilot plant. Stelco carried out the preliminary engineering for such a facility and this information was incorporated into a proposal submitted by Stelco to EMR under the Industrial Energy Research and Development program for support. The review of this application pointed out the lack of a market analysis to support the investment. As a result of the EMR interest in this proposal a EMR/Stelco jointly funded study was proposed to answer the technical and marketing questions necessary.

The jointly sponsored study was undertaken by Stelco under this DSS contract with Hatch Associates Ltd. of Toronto as the main element of the study supervised by Stelco.

Technical Status of the LB Process

The LB Process utilizes fine coal-iron ore mixtures which are fed into,

Statut technique du procédé LB

Le procédé LB utilise des mélanges fins de minerais de fer et de charbon qui sont introduits dans un tube à cornue de carbure de silicium et acheminés à travers le tube au moyen d'une vis. Du monoxyde de carbone, généré par la réaction du charbon avec les minerais de fer et par la réaction Boudouard, est utilisé pour la réduction de masse. Le gaz passe dans le même sens le long du tube et brûle au tube de sortie. Ensuite les gaz chauds repassent en sens inverse à l'extérieur du tube de carbure de silicium pour fournir de la chaleur pour le procédé. Le matériau spongieux réduit, dont le taux de métallisation varie selon le débit de déplacement dans le tube, tombe dans une chambre où il est fondu au moyen d'électricité. Les dimensions du tube expérimental sont de 0,08 m sur 0,75 m et la zone de température étudiée variait de 1000°C à 1200°C. La métallisation variait de 60% à 90% (chiffres estimés). Les taux de production variaient de 40 kg/m²/h à 80 kg/m²/h quand la capacité de traitement est exprimée en termes de la surface disponible pour le transfert de chaleur.

Concept de tubes à cornue multiples

L'objectif précisé pour une première installation commerciale était de 300 000 tonnes par an. Employant les données de l'inventeur et l'hypothèse de base qu'il n'y aurait aucune résistance interne au transfert de chaleur, le taux de réaction de 80 kg/m²/h a été utilisé pour décider de la taille du réacteur. Le choix du tube de carbure de silicium de 0,45 sur 4,1 m entraînerait l'emploi de 75 tubes pour produire les 300 000 tonnes. Il faudrait que les tubes fonctionnent à un facteur d'utilisation de 100%; lorsqu'on tient compte du débit des gaz de

and conveyed through, a silicon carbide retort tube by a screw. Carbon monoxide gas, generated by reaction of the coal with the iron ore and by the Boudouard reaction, is used for the bulk reduction. The gas flows concurrently along the tube and is combusted at the exit tube. The hot gases are then passed countercurrently along the outside of the silicon carbide tube to provide heat for the process. The reduced sponge, of varying metallization depending upon rate of travel in the tube, drops into a chamber where it is melted using electricity. The experimental tube dimensions are 0.08 m by 0.75 m and the temperature range investigated varied from 1000°C - 1200°C. Metallization varied from an estimated 60 to 90 percent. Rates of production varied from 40 $kg/m^2/hr$ to 80 $kg/m^2/hr$ when the throughput per hour is expressed in terms of the area available for heat transfer.

Multiple Retort Tube Concept

The stated aim for a first commercial plant was 300 000 tones per year. Using the inventor's data and prime assumption that there would be no internal resistance to heat transfer, the 80 $kg/m^2/hr$ reaction rate was used to size the reactor. Utilizing a silicon carbide tube size of 0.45 m x 4.1 m, 75 tubes would be required to produce the 300 000 tons. These tubes would have to operate at a 100 percent utilization factor and when allowance was made for heating gas flow between the tubes, this leads to an 8 m x 8 m reactor face. Chunks falling off this face into a melting bath present very difficult problems of electrode placement and splash. In order to minimize the height, and to improve maintainability, a two unit chamber, each unit containing 40 tubes, was envisaged. An

chauffage entre les tubes, il en résulte une façade de réacteur de 8 m sur 8 m. Des gros morceaux qui tomberaient de la façade dans un bain de fonte présentent des problèmes très difficiles quant au placement des électrodes et aux éclaboussements. Pour minimiser la hauteur et pour améliorer les possibilités de maintenance, une chambre à deux unités dont chaque unité contiendrait 40 tubes a été envisagée. Un avantage était qu'une unité serait fermée pour maintenance et réparation pendant que l'autre continuerait de fonctionner. Même avec le concept de deux unités, nous considérions que les dimensions des chambres, la hauteur de chute des gros morceaux, l'alimentation de 40 tubes et même le chauffage de chaque chambre présentaient plusieurs problèmes opérationnels. Le concept de tubes multiples horizontaux comme base d'une installation commercialle a donc été jugé inacceptable et des recherches supplémentaires sur ce concept ont été abandonnées.

Approche par puits annulaire

Pour surmonter les inconvénients du tube multiple, le professeur Lu a suggéré une approche par puits annulaire de grande taille. Les gaz chauds passeraient le long des surfaces extérieure et intérieure de deux tubes concentriques à grand diamètre et aux parois minces, faits de carbure de silicium. Dans l'anneau entre les deux tubes, le mélange fin de minerais de fer et de charbon serait poussé vers le bas par une hélice à pas fin. L'anneau serait de 0,3 m et les diamètres des tubes de l'ordre de 3,5 m et 4,0 m. Le système fonctionnerait à 1500 - 1700°C. Comme chez le tube à cornue horizontal à échelle de laboratoire, le gaz brûlerait à l'extrémité (inférieure) de sortie, mais, à l'encontre du cas du tube horizontal, le gaz brûlerait avec de l'air préchauffé et serait

advantage was that one unit would be down for maintenance and repair while the other operated. Even with the two unit concept, chamber size, height of fall for the chunks, feed to forty tubes and even heating of each chamber presented, in our judgement, several operating problems. This horizontal multi tube concept for a commercial plant was therefore deemed unacceptable and further work on this concept was abandoned.

Annular Shaft Approach

To overcome the multiple tube drawbacks, Professor Lu suggested a large annular shaft approach. Two concentric large diameter thin-walled silicon tubes would have the hot gas passing up their outer and inner surfaces. In the annulus between the two tubes, the fine ore-coal mixture would be driven down by a fine pitch helix. The annulus would be 0.3 m and the tube diameters in the order of 3.5 and 4.0 m. The system would operate at 1500 - 1700°C. As in the horizontal bench scale retort, the gas would be combusted at the exit (lower) end but, unlike the horizontal tube, it would be combusted with preheated air and distributed to heat the inner and outer annulus walls.

The sponge material would fall into a shaft and be heated and melted both by the combusted gas and by electrical power. The aim would be for the sponge to leave the annulus at some 60 percent metallization.

Some preliminary static experiments were run at intermediate temperatures to define possible operating rates. However the experiments have not reached a stage where data sufficient to establish large pilot plant

distribué pour chauffer les parois intérieure et extérieure de l'anneau.

Le matériau spongieux tomberait dans un puits pour se faire chauffer et fondre par les gaz brûlés et de l'éléctricité. Le but serait que le matériau spongieux quitte l'anneau à un taux de métallisation de quelque 60%.

Quelques expériences statiques préliminaires ont été menées à des températures intermédiaires pour définir des taux d'opération possibles. Cependant les expériences n'ont pas encore atteint un stade où elles peuvent fournir des données suffisantes pour établir des paramètres adéquats pour une installation pilote de grande taille. Des domaines de préoccupation possibles qui exigent du travail expérimental comprennent: la confirmation du mécanisme du transfert de chaleur et des taux de réaction comme fonction de l'épaisseur de la couche; l'établissement de réactions éventuels (et de leurs taux) entre les réductants et les parois des tubes de carbure de silicium soumis à des températures élevées; les effets de fontes locales aux parois chaudes sur les questions de la perméabilité de la couche et du transfert de chaleur; l'analyse des techniques de bourrage nécessaires pour neutraliser la tendance à se gonfler que montre le mélange de réactant dans l'anneau du puits à des températures intermédiaires; et l'examen de la quantité de carbone excédentaire exigée pour empêcher que la masse du matériau spongieux dans le puits ne se réoxyde au contact des gaz brûles de déchet.

Il y a aussi nombre de problèmes du génie dont la solution dépasse le travail expérimental, y compris la stabilité des tubes de puits de

parameters have been established. Possible areas of concern that require experimental work include: confirmation of the heat transfer mechanism and rates of reaction as a function of bed thickness; establishment of possible reactions and rates between the reductants and the silicon carbide tube walls at elevated temperature; effects of localized melting at the hot walls on bed permeability and heat transfer; determination of packing techniques to overcome the swelling characteristics of the reactant mixture at intermediate temperatures in the shaft annulus; and examination of the amount of excess carbon required to prevent reoxidation of the sponge mass in the shaft as it is exposed to the combusted waste gas.

There are also a number of engineering problems whose solution extends beyond the experimental work. These include the stability of the silicon carbide shaft tubes, particularly when in tension from the action of the screw and the swelling of the reactants. The structure of the top driven helix and its stability is not easily solved. Feeding material in and out of the annulus, and efficient combustion and distribution of the reactant gas pose significant problems.

Marketing

The four marketing reports which complement the technical reports for the LB Process study are:

1. World Steel Trends
2. Review of DRI/Liquid Ironmaking Processes
3. Summary Review of World Energy Trends
4. EAF Dust Treatment

carbure de silicium, surtout lorsqu'ils sont mis sous tension par l'action de la vis et du gonflement des réactants. La structure de l'hélice actionnée par le haut et sa stabilité posent un problème qui n'est pas facile à résoudre. Introduire des matières dans l'anneau, les décharger de l'anneau, brûler et distribuer le gaz de réactant de façon efficace, voilà encore des problèmes de taille.

Marketing

Les quatre rapports de marketing qui complètent les rapports techniques pour le procédé LB sont les suivants:

1. Tendances mondiales de l'acier
2. Revue des procédés de fabrication de fer DRI/liquide
3. Résumé des tendances mondiales de l'énergie
4. Traitement des poussières de four électrique à arc

Aucune évaluation du taux de rentabilité éventuel du procédé LB n'a été possible, même en ce qui concerne son utilisation pour l'élaboration d'unités de fer à partir de minerais et de charbon, car au cours de l'évaluation technique il est devenu évident que des recherches supplémentaires étaient nécessaires pour faire face à divers paramètres de procédés importants. De même les recherches sur le procédé LB en ce qui concerne le traitement des poussières de four électrique à arc ont été retardées, en raison surtout du besoin de modifier l'équipment; des calculs du taux de rentabilité faits à l'heure actuelle n'auraient ainsi qu'une valeur limitée.

Le rapport "Tendances mondiales de l'acier" donnait un survol qui a permis une évaluation préliminaire de la possibilité que le procédé LB

No assessment of the potential return on investment (ROI) of the LB Process has been possible, even for its use for the production of iron units from ore and coal, as it became evident during the technical evaluation that further research was necessary to address various important process parameters. Likewise the research of the LB Process for EAF dust treatment has been held up, due primarily to the need for equipment alterations and any ROI calculations at this time would be of limited value.

The "World Steel Trend" report gave an overview which allowed a preliminary assessment for the possibility for the LB Process finding a place in future ironmaking decisions. In general it was concluded that the current and future steel market has a need for iron units which can be produced by some direct reduction process, either resulting in DRI sponge or liquid iron. This production requirement is most likely where there is a need for relatively small increments of capacity.

The LB Process selection would be enhanced where there is low-cost coal (but not of coking grade), no low-cost natural gas, a shortage of economically available scrap and moderately priced electricity. It is suggested that in the areas of the world which are expected to undergo most change in steelmaking facilities (developing countries, China, and to some extent the CPE countries) there is likely to be a greater opportunity for DRI and the LB Process.

soit pris en compte dans des décisions futures sur l'élaboration du fer. En général la conclusion était que le marché actuel et futur de l'acier montre un besoin d'unités de fer qui peuvent être élaborées par un procédé de réduction directe qui aboutit à du matériau spongieux des DRI ou au fer liquide. Cette exigence d'élaboration a le plus de chances d'exister là où il y a le besoin d'augmentations de production qui soient relativement faibles.

Le choix du procédé LB serait plus probable là où il existe du charbon à bas prix (mais qui ne soit pas de grade cokéfiable), aucun gaz naturel à bas prix, une carence de ferrailles disponibles à prix économique et de l'électricité à prix modéré. On suggère qu'il y aura vraisemblablement des possibilités accrues des DRI et du procédé LB dans les parties du monde où sont attendus les changements les plus importants dans les aciéries (les pays en voie de développement, la Chine, et dans une certaine mesure les pays de l'économie du Pacifique central).

Le rapport de la "Revue des procédés de fabrication de fer DRI/liquide" passait en revue la multitude de procédés de réduction directe capables de produire soit des DRI soit du fer liquide. Les classifications des procédés par type de minerais d'alimentation (fines ou agglomération, soit en agglomérés soit en boulettes), par forme d'énergie de réduction, et par type de réacteur mettent en valeur le fait qu'il n'existe pas à l'heure actuelle de procédés commerciaux des DRI qui utilisent une chambre de chauffage indirect comme chez le procédé LB. Deux procédés commerciaux qui ressemblent au procédé LB en ce qu'ils emploient des fines de charbon et de minerais, sont le système de New Zealand Steel

The "Review of DRI/Liquid Ironmaking Processes" report considered the multitude of direct reduction processes which can produce either DRI or liquid iron. Classifications of the processes by types of ore feed (fines or agglomerated, either sinter or pellets), by reductant energy form, and by reactor type points out that there are currently no commercial DRI processes which use an indirect heating chamber as in the LB Process. Two commercial processes which are similar to the LB Process, in that they use coal and ore fines, are the New Zealand Steel operation (based on SL/RN kiln, ironsands and electric reduction furnace to provide liquid iron) and the Elred process. If opportunities for the LB Process are in developing countries it is important that the equipment and process not be overly complex in order to attain a high operating performance with relatively unskilled personnel.

The developments aimed at thin steel slabs being continuously cast are likely to cause a major revolution in the steel industry if fully successful. Flat-rolled steel products can be made via this route. Thus the massive and expensive hot-rolling breakdown mill and significant slab surface treatment and other processes can be eliminated or reduced. The resulting ability for the small mini-mills, based on the electric arc furnace, to then compete with the large Blast Furnace/BOF installations would drastically alter the feed requirements. To produce the higher quality flat-rolled products would require more virgin iron units in the feed and less scrap steel (which introduces undesirable tramp elements). Also an increase in the overall tonnage requirements for electric arc furnace produced steel might put some pressure on scrap supplies and hence raise the scrap price.

(basé sur un four SL/RN, des sables ferrugineux et un four de réduction électrique pour fournir du fer liquide) et le procédé Elred. Si des occasions pour le procédé LB existent dans les pays en voie de développement il est important que les installations et le procédé ne soient pas excessivement complexes afin d'atteindre une performance fonctionnelle élevée avec un personnel relativement peu qualifié.

Il est probable que les développements visant à la production de brames minces d'acier en coulée continue causeront une révolution majeure dans l'industrie de l'acier si les développements réussissent pleinement. Des produits d'acier laminés à plat peuvent se fabriquer de cette façon. Ainsi seraient éliminés ou réduits l'usine massive et coûteuse de laminage à chaud incorporant un stade de laminage intermédiaire de brames, le traitement important de surfaces de brames et d'autres procédés. Les mini-usines de petite faille à base du four électrique à arc acquerraient par conséquent la capacité de concurrencer les implantations de grante taille de type haut fourneau/aciérie à l'oxygène, ce qui aurait pour effet de changer de façon dramatique les exigences en alimentation. La fabrication des produits laminés à plat de qualité supérieure exigerait une alimentation à plus forte concentration d'unités de fer vierge et avec moins de ferrailles (qui introduisent des éléments indésirables). Par ailleurs, l'augmentation de la demande totale de tonnes d'acier produit en fours électriques à arc pourrait exercer de la pression sur les stocks de ferrailles et faire ainsi monter le prix de la ferraille.

Le rapport du "Resumé des tendances mondiales de l'énergie" examinait les

The "Summary Review of World Energy Trends" report examined the world's known energy resources to determine where it is likely that a coal-based process such as the LB Process would have some advantages, i.e. where there is relatively cheap coal and no available low-cost natural gas.

The four major exporters of hard coal have been Australia, USA, South Africa and Poland, followed by Canada and the USSR. Increased exports are expected by South Africa, China and Columbia in the 1990's. All factors being equal, these exporting countries are assumed to be sources of abundant and inexpensive coal and thus likely to be favorable sites for the LB Process. Other locations where transportation costs for the coal to the steel producing operations are low are also potential site locations.

A comparison of coal production and natural gas production in selected steelmaking countries revealed that coal was the dominate energy source in Brazil, China, India, South Africa, North Korea and South Korea. Again there may be some advantage in these countries for the LB Process.

Forecasts for a higher share of the world's primary energy to be produced from coal coupled with the expectation that the price for coal will be relatively cheaper compared to natural gas and oil would be one factor favoring the future production of direct reduction iron units using coal as a reductant rather than natural gas.

The "EAF Dust Treatment" report was not able to address the potential for the LB Process for this use in any meaningful way. However it does

ressources énergétiques mondiales connues pour décider des sites où il est le plus probable qu'un procédé à base de charbon, tel que le procédé LB, aurait des avantages, c'est-à-dire où il existe du charbon relativement bon marché mais aucun gaz naturel disponible à bas prix.

Les quatre exportateurs majeurs de houille anthraciteuse ont été l'Australie, les Etats-Unis, l'Afrique du Sud et la Pologne, suivis du Canada et de l'URSS. Des exportations accrues sont attendues dans les années 1990 en provenance de l'Afrique du Sud, la Chine et la Colombie. Dans l'hypothèse que tous les facteurs soient égaux, l'on suppose que ces pays exportateurs sont des sources de charbon abondant et peu coûteux, et qu'il est probable qu'ils sont des sites favorables au procédé LB. D'autres endroits où les coûts de transporter le charbon jusqu'aux aciéries sont peu élevés constituent également des sites possibles d'installations.

La comparaison de la production du charbon et du gaz naturel dans certains pays producteurs d'acier a révélé que le charbon était la source d'énergie dominante dans les pays suivants: Le Brésil, la Chine, l'Inde, l'Afrique du Sud, la Corée du Nord, la Corée du Sud. Encore it se peut qu'il y ait dans ces pays des avantages pour le procédé LB.

Des prévisions qu'une part plus grande de l'énergie primaire du monde viendront du charbon, jointes aux attentes que le prix du charbon sera relativement plus bas en comparaison avec celui du gaz naturel et du pétrole, constitueraient un facteur en faveur du la production future d'unités de fer au moyen de la réduction directe utilisant comme

appear from the information coming from the USA Environmental Protection Agency (EPA) that processes will be favored which can recover for reuse most of the contained materials in the dust. Those processes which satisfy the environmental requirements for safe disposal without reuse of the metals value will be somewhat less desirable.

The LB Process does offer the potential to recover most of the metal values and to have, as a disposal residue, a non-hazardous slag. The same potential, however, also exist for other EAF dust treatment processes which are further advanced. Timing is obviously important as the removal of landfilling as an option for electric arc furnace steelmaking facilities' disposal of EAF dust is rapidly approaching in the USA. The intended deadline of August 8, 1988 may be extended, but only for up to two years. There will remain some scope for exceptions but this is likely to be restricted to no more than about 10 percent of the EAF dust being produced. The LB Process suffers from this timing factor but there is and will be a continuing need for research and development aimed at finding a better technical and economic solution to the EAF dust treatment problem. Thus, it is of value for research on the LB Process to continue as long as it holds promise.

Conclusions

1. Further laboratory research is required to more clearly define the process parameters before the question of technical success can be determined.

réductant le charbon plutôt que le gaz naturel.

Le rapport du "Traitement des poussières de four électrique à arc" n'a pu traiter d'une façon significative le potentiel qu'aurait le procédé LB de s'employer à cette fin. Il paraît cependant, d'après les renseignements fournis par l'Agence de la protection de l'environnement des Etats-Unis ("EPA") que seront favorisés les procédés qui peuvent récupérer pour réutilisation la plupart des matières contenues dans les poussières. Les procédés qui satisfont aux exigences environnementales au sujet de la décharge sécuritaire sans réutilisation de la valeur des métaux seront moins à désirer.

Il est vrai que le procédé LB offre la possibilité de récupérer la plupart des valeurs des métaux et d'avoir, comme résidu de rebut, des scories non-nocives. Cependant le même potentiel existe également pour d'autres procédés de traitement de poussières de four électrique à arc qui sont plus avancés. Le rhythme des démarches est d'une importance évidente puisqu'aux Etats-Unis il ne sera bientôt plus possible de recourir à l'option du remblai pour disposer des poussières de four électrique à arc produites par les aciéries électriques. La date-limite prévue peut être reportée mais non pas au delà de deux ans. Il restera la possibilité de quelques exceptions mais elles ne toucheront vraisemblablement pas plus de 10% environ des poussières produites par les fours électriques à arc. Le procédé LB souffre de ce facteur de chronologie; cependant il reste--et il restera--le besoin permanent de recherches et de développement visant à trouver une meilleure solution technique et économique au problème du traitement des poussières de four

2. Market analysis indicates that a coal based ironmaking process has a potential market in some specific areas of the world.

3. If the LB Process can handle EAF dust and other iron bearing wastes in a manner that the off gas by-products are in a reusable form, economically a market for this specific purpose will most likely exist as no other process is yet processing this material economically in an environmentally acceptable fashion.

Recommendations

1. That the McMaster University team be encouraged to continue the development of the LB Process on a laboratory scale to define the process in a vertical mode.

2. That experiments be carried out to verify the heat transfer mechanisms at the temperatures and under the conditions that are predicted for the commercial process.

3. That the technical assessment of the LB Process be reviewed when the experimental data is complete to determine the process' commercial viability.

G. A. Chapman

GAC:CO
December 29, 1987

électrique à arc. Ainsi les recherches sur le procédé LB méritent de se poursuivre aussi longtemps qu'elles continuent de promettre.

Conclusions

1. Des recherches plus poussées en laboratoire sont exigées pour définir plus clairement les paramètres du procédé avant que la question du succès technique ne puisse se décider.
2. L'analyse du marché indique qu'un procédé d'élaboration du fer à base de charbon a un marché éventuel dans certaines parties spécifiques du monde.
3. Si le procédé réussit à traiter les poussières de four électrique à arc et d'autres déchets ferreux de sorte que les sous-produits contenus dans les gaz de déchet sont présentés dans une forme réutilisable, il est très probable qu'un marché économique existera, vu qu'aucun autre procédé ne traite encore ces matières d'une façon à la fois économique et acceptable sur le plan environnemental.

Recommendations

1. Que l'équipe de l'Université McMaster soit encouragée à continuer de développer le procédé LB à l'échelle de laboratoire pour définir le procédé dans un mode vertical.
2. Que se fassent des expériences pour vérifier les mécanismes de transfert de chaleur aux températures et sous les conditions prévues pour le procédé commercial.

3. Que l'évaluation technique du procédé LB soit revue quand les données expérimentales seront complètes pour déterminer la viabilité commerciale du procédé.

G. A. Chapman

GAC/NL
December 29, 1987

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LB FURNACE STUDY

FAMILIARIZATION AND QUANTIFICATION

P.R. #5006.1 Rev.1
File #5006.2

October 22, 1987

PROJECT REPORT

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1.0 SUMMARY

Utilizing the bench scale data developed at McMaster University, a suggested industrial configuration has been developed for the LB process. An initial plant size of 300,000 tonnes per year (tpy) was the aim. Using optimistic heat transfer assumptions, the required area for heat transfer and the limitation on retort size led to a 75 tube reactor assembly. In order to decrease size, the plant was broken into two 150,000 tonne units. Plant complexity and the uncertainties of scale-up from the bench lead to an operating time factor estimate of 0.67 for the first plant, and to a projected capacity of 200,000 tonnes per year, with 300,000 tpy achievable with successful process development.

The multitube reactor approach faces serious material and heat transfer problems. For the liquid iron step, production of large sponge chunks in the larger reactor tubes is unfavourable for direct charging and melting.

2.0 INTRODUCTION

The LB, a two stage direct reduction-melting process, currently is a bench-scale operation. This report considers the factors of heat transfer, process temperatures and degree of metallization for the operation. Based on laboratory results, a preliminary analysis of process scale-up to commercial size is presented. Possible configurations for the direct reduction and the melting units are suggested.

2.1 Existing Process

The LB Furnace in the bench-scale consists of two components, one for production of sponge iron and the other for melting (Fig. 1). Sponge iron is produced in a tubular reactor. Raw materials at ambient temperatures are fed to the reactor and are moved through the tube by a steel screw feeder. Hot gaseous products (mainly CO) from the reaction tube protect the sponge iron from reoxidation and, when combusted at the end of the tube, provide the heat necessary for the reaction. The sponge iron leaves the silicon carbide tube at the hot end, breaks off and falls into the bath where it is heated and melted by plasma.

2.2 Scale-Up to a Commercial Process

Scale-up to a commercial process will result in a much more complex operation than the bench-scale. Of primary importance will be heat transfer, size and configuration of the commercial unit. Capacity requirements for the first commercial plants are discussed. The scale-up basis and a potential configuration for the first commercial unit is defined. Probable operating difficulties and possible solutions are outlined.

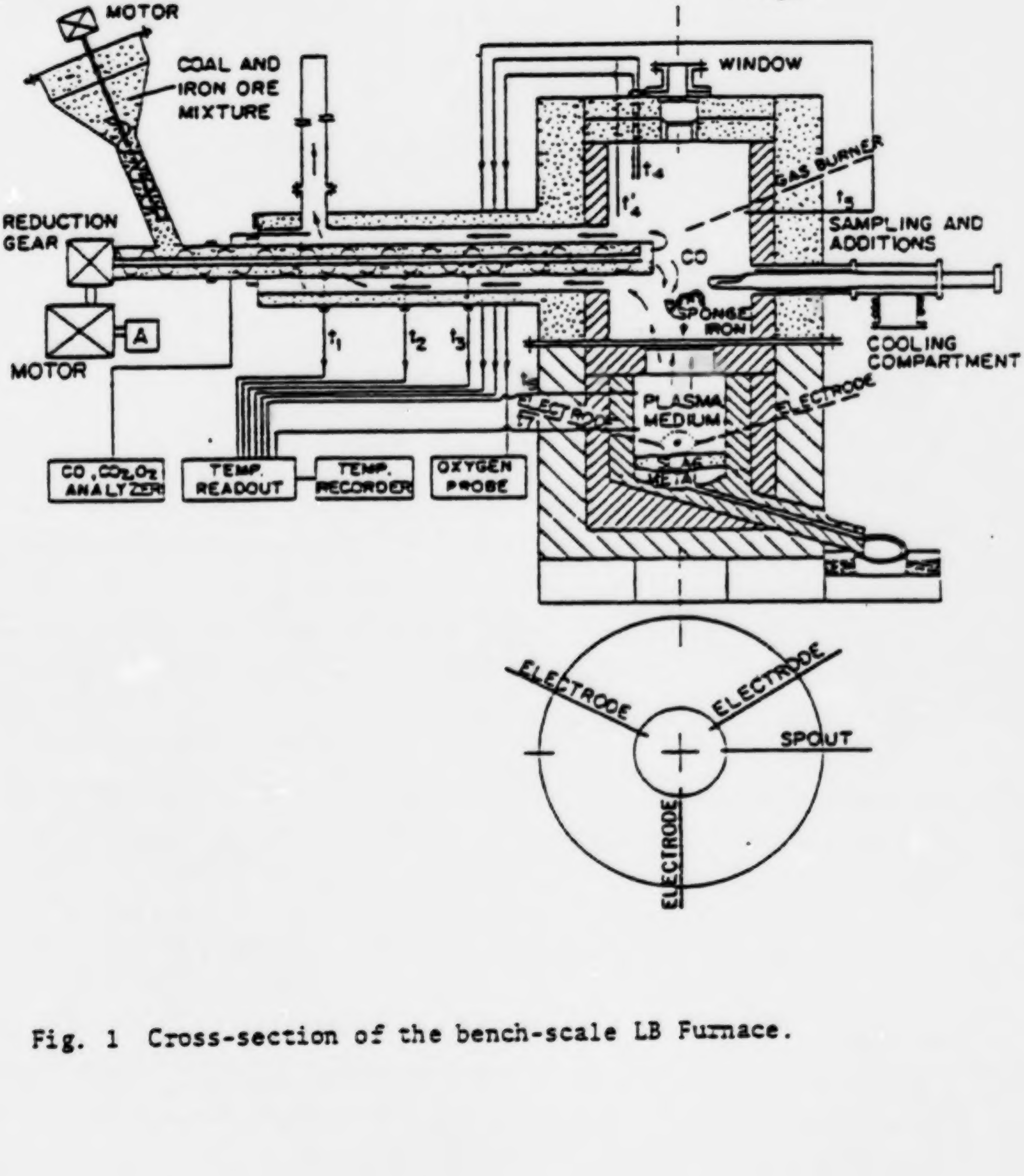

MOTOR
COAL AND
IRON ORE
MIXTURE
WINDOW
GAS BURNER
t_4
t'_4
t_5
REDUCTION
GEAR
SAMPLING AND
ADDITIONS
CO
SPONGE
IRON
A
MOTOR
t_1
t_2
t_3
COOLING
COMPARTMENT
t_6
ELECTRODE
PLASMA
MEDIUM
ELECTRODE
SLAG
METAL
CO, CO2, O2
ANALYZER
TEMP.
READOUT
TEMP.
RECORDER
OXYGEN
PROBE
ELECTRODE
ELECTRODE
SPOUT
ELECTRODE

Fig. 1 Cross-section of the bench-scale LB Furnace.

3.0 CAPACITY REQUIREMENTS FOR FIRST TWO PLANTS

If new ironmaking processes are to succeed, they must exceed some minimum annual production level. For direct reduction plants today, this level must be influenced to some extent by the capacity of the most important commercial processes. Third generation Midrex gas reduction modules are rated to 600,000 tonnes/year with sizes ranging from 200,000 tonnes/year. A 300,000 tonnes/year Corex (KR) unit for coal-based reduction to liquid iron is being built in South Africa, after experience with 6 tonne/hour pilot plant. Sumitomo has run elements of their small pilot plant for coal based ironmaking at 18 tonnes/day and have visualized a 100 tonne/day pilot plant as the next step before proceeding to commercial units of 365,000 tonnes/year and 900,000 tonnes/year. The LB process must compete with these coal and iron ore to liquid metal processes which have already reached first commercial or demonstration plant size.

It is our opinion that the first two plants, if they are to be considered as primary components in the iron and steel industry, should have annual capacities of:

Plant 1: 300,000 tonnes/year
Plant 2: 600,000 tonnes/year

Although this report does not consider electric arc furnace dust treatment facilities, they fill a different commercial need and small capacities become practical. For initial purposes, we suggest that dust treaters have capacities of:

Plant 1D: 15,000 tonnes/year
Plant 2D: 50,000 tonnes/year

Dust treatment provides an opportunity to get into the business on a smaller scale and for much less money than would be required for primary iron production. (Using the dust treater as a step on the road to a primary iron plant would have a longer development cycle, but has positive aspects, in particular, a small scale-up from pilot plant to commercial dust treatment facility).

3.1 The Reduction Unit

It is intended that the Lakeview pilot plant have one reactor tube, 0.30 m internal diameter. Armed with this experience and data, it is reasonable to envisage an increased tube size for the commercial plants. We propose, therefore, that the commercial plants might have tubes as follows:

Plant	Tube Size	Capacity
Plant 1:	0.45 m x 4.1 m	300,000 tpy*
Plant 2:	0.60 m x 5.4 m	600,000 tpy
Plant 3:	0.90 m x 8.1 m	600,000 tpy

Tubes are to be silicon carbide, wall thickness to be established.

3.2 Degree of Metallization

Experimentally the LB process has operated over a range of metallization from 60% to 95%. The degree of metallization can be controlled by adjusting the screw feed and varying the unit production rate or temperature. The degree of reduction of the sponge dropping to the melting zone is a very important parameter as it determines the melting energy requirement, the dust build-up and the possibility of clean gas combustion.

*NOTE: This is a two unit plant. The rationale will be developed below.

In the LB process high and low metallization have their advantages and disadvantages as outlined below.

3.2.1 High Metallization

Pros:

- sintering of the material allows large driving forces to be used for heat transfer and results in dust-free sponge, allowing for high temperature combustion of reactant gases without gas cleaning
- smaller electrical load and electric energy consumption on melting, hence lower energy costs
- possibility of passing directly to steelmaking
- most broadly innovative

Cons:

- lower productivity in the reduction unit
- difficult transfer to melting, perhaps to the point that batch transfer is necessary
- load and wear on the screws
- load and wear on the tubes, particularly at the high stress break-off zone
- ability to recover from interruptions
- unknown stability of the chunk size

3.2.2 Low Metallization

Pros:

- transfer into, and control of, melting relatively straightforward
- steady dribble discharge
- increased productivity of reduction unit
- predictable load on screws

Cons:

- dusty gas may need cleaning before burning, based on our experience in this area
- stability of the carbon-oxygen balance in feed to the melter may be poor, especially after operating interruptions or delays
- higher electrical energy input will increase overall energy costs
- sintering may occur during interruptions

3.3 Temperatures

The laboratory work has used temperatures of 1050°C to 1250°C close to the reduction tube. Most work was carried out at 1200°C.

This is a high temperature in direct reduction process terms in that some components of the charge will melt and the sponge mass will sinter to some extent. Melting and sintering may be desirable, as recognized by the developers, in order to minimize dust expulsion on transfer to melting. However, if sintering is excessive, the material produced may be too strong or too large, and will present a problem at the break-off point in the reactor. Also, in this temperature range sintering and coalescence has contributed to a decrease in the overall reaction rate in other sponge processes.

The developers feel that a temperature of 1400°C is attainable at the tube wall's hottest spot, thus greatly increasing the radiative heat transfer to the sponge. This poses no short term problems for the silicon carbide tube, but may be a problem if the screw feeder is exposed. This temperature will result in increased metallization and additional melting and sintering at the tube exit compared to operation in the 1050-1200°C range.

In "Notes on Scale-Up and Heat Transfer" by W.K. Lu (April 17, 1986), the bench-scale work shows the progress of the charge at 0.08 m/min. The production rate is reported as 77 $kg/m^2/hr$ of iron and the expectation for the Lakeview facility is 98 $kg/m^2/hr$, based on using a 25% higher packing density of the reactant mixture than the bench unit. Projection to commercial scale then predicts 100 $kg/m^2/hr$, 200 $kg/m^2/hr$ by raising the temperature to 1400°C and 260 $kg/m^2/hr$ by altering the configuration exposed to the combustion chamber at 1400°C.

For preliminary calculation purposes for the reduction unit in Plant 1, a value of 80 $kg/m^2/hr$ is used. To size the units in Plants 1, 2 and 3, values of 80, 100 and 200 $kg/m^2/hr$ are used, respectively.

3.4 Scale-Up in the Reduction Unit

The most significant of the scale-up factors will be heat transfer through the tube walls. This is a surface area and thickness factor. Other factors will affect heat transfer rates but they will be ignored in this analysis. (One of these, is transfer or diffusion of heat throughout the mass of material inside the tube. Ignoring this will give a best-case scenario.)

A scale-up factor that recognizes the heat transfer limitation is the production rate expressed as mass of iron per hour per area of the tube wall in the furnace.

An immediate impression of the magnitude of this number can be had from Paper No. 3, Fig. 8. In a 0.083 m ID and .75 m long silicon carbide tube the allowable speed of the charge was 0.03 m/min. at 90% metallization. This yielded a production rate of 35 $kg/m^2/hr$ of iron.

3.4.1 The Reduction Unit in Plant 1 (Preliminary Calculations)

For Plant 1, the desired iron production rate is 300,000 tonnes/yr., about 34 tonne/hr., and the required tube wall area in the furnace would be 34,000/80 = 425 m^2. The practical length of a tube is related to its diameter. The screw will have the equivalent of a columnar load imposed on it. A screw shaft diameter to length ratio of 30 may be just practical; if the screw shaft to tube diameter ratio is 0.3, we have a length to diameter ratio for the tube of 9 (Lu has used 8.5). The following illustrates the approximate geometries of the reactor and combustion chamber face if tubes are placed on a 2 x diameter (2D) square array:

Reactor Configuration for 300,000 Tonnes Per Year

Tube Internal Diam (m)	Tube Length (m)	Surface Area (m^2)	Number of Tubes	Next Whole Number Per Side	Dimension of Reactor Face (mxm)
0.10	0.90	0.28	1518	39	7.8 x 7.8
0.15	1.35	0.64	664	26	7.8 x 7.8
0.20	1.80	1.13	376	20	8.0 x 8.0
0.30	2.70	2.54	168	13	7.8 x 7.8
0.45	4.05	5.73	75	9	8.1 x 8.1
0.60	5.40	10.18	42	7	8.4 x 8.4
0.90	8.10	22.90	19	5	9.0 x 9.0

A single tube is infeasible. The choice is between a few big or many small tubes. One advantage of small tubes is that the pilot plant could use a few next-plant size tubes, so enormously securing scaleup. This has been done in the chemical industry and with reformers for natural gas-based DRI processes. The reactor face at 8 m x 8 m is so large that it immediately raises the question of whether 300,000 tonnes annually should be accommodated in one unit. At an 8 m height, the fall of some of the sponge into the bath will exceed 12 m, which would pose splash problems. Also, the horizontal spread of the charge will be 8 m, unless it is narrowed by a chute. At 1200°C, with a chunky irregular material, a chute may be inoperable. Possibilities are:

- accept a chute
- accumulate in a bucket for batch transfer to a melter
- use multiple units
- forego hot transfer

3.5 Number of Reduction Tubes

The number of tubes is a function of heat transfer rates. The inventors experienced a production rate of 35 to 77 kg/m^2/hr of iron in the laboratory and forecast 98 kg/m^2/hr in the Lakeview unit. In commercial units at the same temperatures they project 100 kg/m^2/hr, then 200 kg/m^2/hr by raising the temperature, and finally 260 kg/m^2/hr by altering the geometry. We propose the following protocol based on tube length/diameter = 9:

Plant	Reaction Rate of Iron kg/m^2/hr	Temp °C	Tube Diameter m	Production Annual Tonnes Fe	Minimum No. of Tubes
1	80	1200	0.45	300,000	75
2	100	1200	0.60	600,000	67
3	200	1400	0.90	600,000	15

3.6 Reduction Unit Arrangement

Tubes in the reduction unit could be set in rectangular array with a cell size of two tube diameters. To avoid impingement of falling sponge chunks, and to allow space for mechanisms at the feed end, higher tube rows are displaced axially into the discharge zone by one-half diameter.

Plant	No. of Tubes	Reactor Face Area All Tubes m^2	Tube Array Number Horizontal/ Vertical	Horizontal Discharge Area m x m	Max Vertical Drop m
1	75	61.0	7 x 11	6.3 x 2.4	10
2	67	96.5	7 x 10	6.3 x 2.2	9
2	15	48.6	3 x 5	5.4 x 2.3	9

3.7 Heat Transfer In the Bench Unit

The process heat is supplied by combusting the gas leaving the reaction tube with air. The combusted gas is passed countercurrently over the outside of the reaction tube to provide additional heat for the reaction.

According to calculations provided by the inventor, heat transfer occurs in three zones. Approximately 35% occurs by radiation to the sponge at the tube end, 45% occurs by radiation to the tube wall extending into the combustion chamber and 20% is due to countercurrent flow of the combustion gas over the outside of the reaction tube. (The combustion gas leaves the system at a temperature of 750°C and thus effort to recover or use some of this heat is required.) To date, neither we nor the inventors have considered internal resistance to heat transfer, though in scale-up this could become extremely important.

3.7.1 Heat Transfer in the Commercial Unit

The first commercial unit will consist of a chamber with tube rows displaced axially forward into the primary combustion chamber. Tubes in the upper rows, especially those in the top row, will experience more radiative heat transfer than bottom row tubes and the degree of reduction and metallization in top row tubes will be higher. Tubes on the sides will be exposed to more radiation than centre tubes, shielded by others. Overall, the sponge leaving a reaction tube will differ in metallization based on its location. Potentially, a very non-uniform product would enter the melting zone and it is possible that some product will not be sufficiently reduced. This could lead to further heat transfer problems with dusty gas as the dust will absorb radiation from the combusted gases and make combustion difficult. A possible solution is to run the screw feeders at different rates based on tube location, but it may be difficult to determine the degree of metallization of the sponge in each tube to the degree necessary to control individual screw feeder rates. An indirect method would be to run the screw feeders at rates based on mean heat transfer calculations in order to ensure that the material in all of the tubes receives the same thermal history.

In order to heat the full length of the tubes, it may be better to partially combust the gas in the primary chamber and burn the excess in additional chambers to provide radiant heat and hot combustion gases to the entire bundle. It might also be desirable to have a waste heat recuperator circuit to preheat combustion air or ore fines. This should be examined.

3.8 Structural Considerations

It is assumed that large silicon carbide tubes can be made (this is being determined). Whether it will be necesary to join short sections is not known. Given that multiple tubes will be required for commercial units, one problem is tube support. Also, it is not feasible to draw the combusted gas evenly along the outside of the tubes for any multiple tube reactor chamber. Bridgewalls could be used to provide tube support and the gas could be passed across the tube bundle up to four times, in order to obtain radiative and convective heat transfer to the tube walls. The gas flow and bridgewall siting must be studied in order to provide optimum heat transfer uniformity to the reactor tubes.

The forces acting on the tubes become a factor. In the proposed commercial units flame temperatures of 1650°C will be observed and maximum temperatures on the tube wall will reach 1400°C. As a result, the tensile strength of silicon carbide may not be sufficient to support the loads, especially of the cantilevered tubes in the upper rows. Also, at high temperatures chipping or reaction of silicon carbide with iron may occur at the tube lip where sponge breaks off.

Damaged tube replacement needs development. It may be possible to cut off the tube and allow it to fall to the melting furnace. It may be necessary to use a core drill to remove the tube from the bridgewalls. Tube insertion techniques must be developed.

It is assumed that it will be possible to obtain screw feeders that can withstand the process stresses. The method of support for these screw feeders for long tubes must be developed. For commercial plants this problem could be significant.

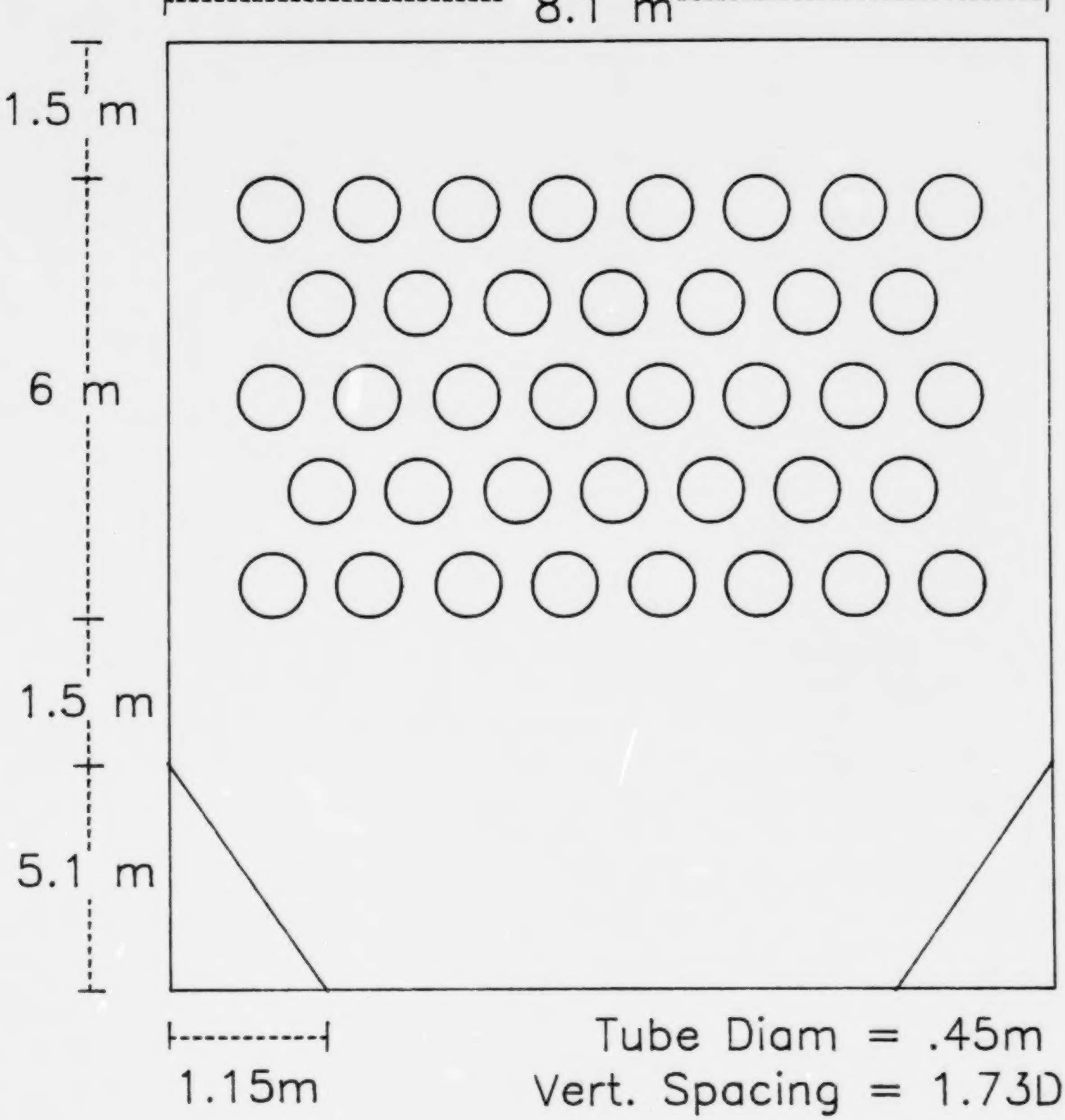
3.9 Commercial Units Visualized

It would be possible to decrease the size of the 300,000 tpy unit by decreasing the tube spacing. Flow of combustion gas around the tubes would have to be investigated. Decreased spacing would not greatly reduce the distance through which the sponge falls into the melting zone and may result in more radiation shielding of the tubes in the centre of the chamber.

For Plant 1, the 0.45 m ID x 4.05 m long silicon carbide tubes are at the size limits of commercial availability. To produce 300,000 tpy 75 tubes are required to operate at 100% availability. Also the reactor face vertical height would approach 10 m even before a gas offtake and a collector chute are considered.

For a first commercial plant, an alternate approach would be to consider two 150,000 tpy units, each of 40 tubes. This would have the added advantage that one unit could operate while the other was down for design improvements, maintenance and repairs. For planning and cash flow purposes one could assume that the operability factor of Plant 1 will be 0.67 because of the many unknowns and complexity of the multitube reactor. This leads to a prediction of 200,000 tpy production from two units, with a potential to achieve 300,000 tpy after successful process development.

A configuration for the 150,000 tpy unit is shown in figures 2 and 3. The tube chamber consists of 5 rows of 7 and 8 tubes. The horizontal spacing is 2D while the vertical spacing is 1.73D. The upper row extend approximately 2 m further into the combustion chamber than the bottom. The maximum height through which sponge must fall to the collector chute is 9.5 m, including the height of the gas offtake. Partial combustion takes place in the primary combustion chamber and combustion is completed in the secondary combustion chambers. Combustion air is introduced through tubes in the wall opposite the tube exits.


8.1 m
1.5 m
6 m
1.5 m
5.1 m
1.15m

Tube Diam = .45m
Vert. Spacing = 1.73D
Horiz. Spacing = 2.0D

Fig.2 LB Furnace Configuration
100,000 t/y Unit

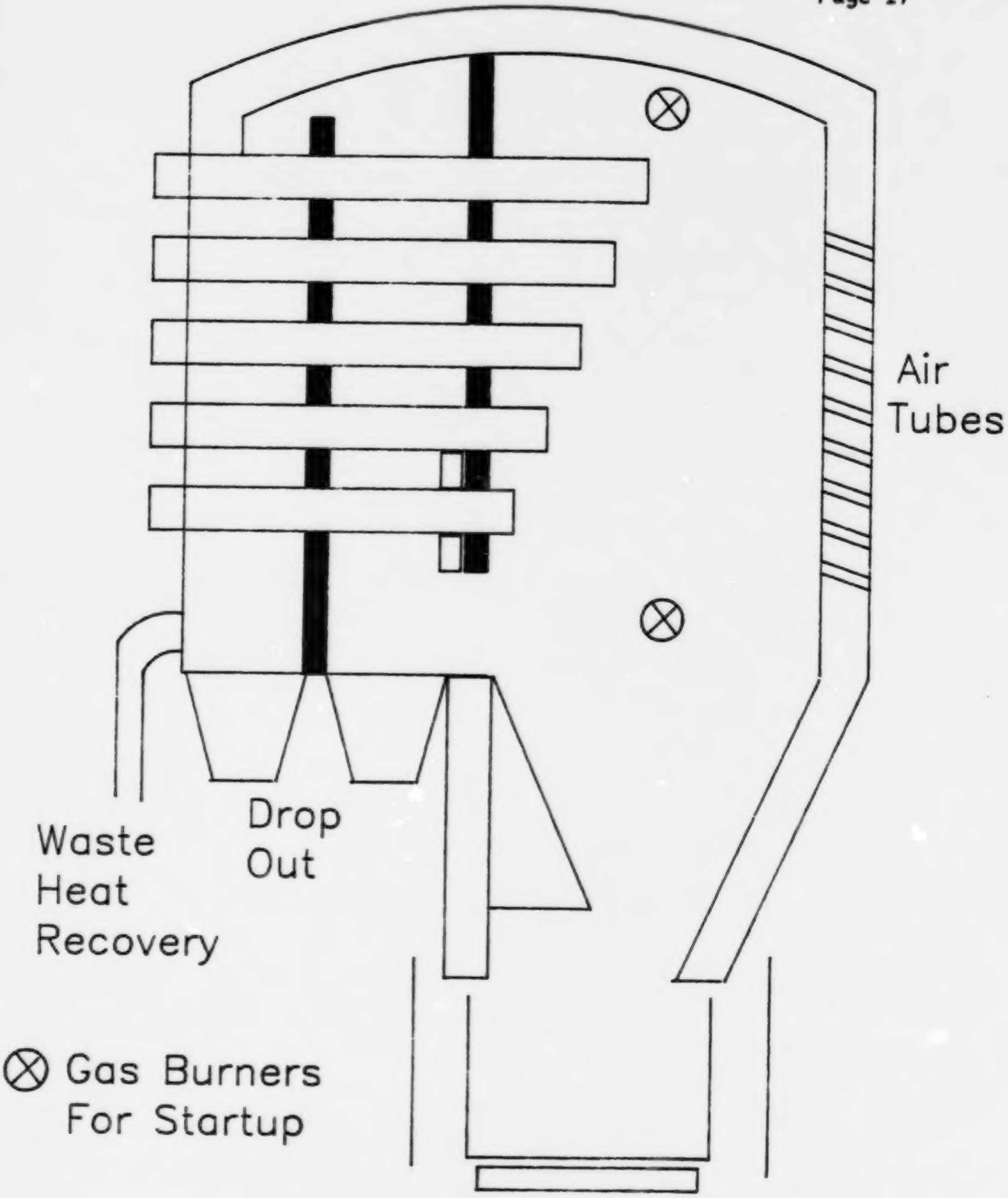

Air
Tubes
Waste
Heat
Recovery
Drop
Out
⊗ Gas Burners
For Startup

Fig.3 LB Furnace Schematic
100,000 t/y Unit

4.0 THE MELTING UNIT FOR A PRELIMINARY PLANT

4.1 Governing Factors

The process of transferring reduced material to the melting unit entails many restrictions arising from the reactor geometry, the material, and from the melting unit demands.

4.1.1 Demands of the Melting Unit

(1) For Plant 1, something around 5 MW of applied power per unit.

(2) Reasonable stability in the ratio between material input rate and power. Variations will lead to temperature swings in the melt. At one extreme, it results in freezing of the bath; at the other sudden gas evolution, foaming and refractory problems.

(3) Some length of furnace campaigns. The steel furnace approach in which minor refractory rehabilitation follows every heat could be adopted, but would lead to complications in the reduction unit (start-stop operation).

(4) A feeding pattern related to the location of energy release. In a three phase furnace, all three electrodes must absorb some feed; otherwise slag and sponge will be overheated in some areas, in others it will freeze. Feeding far away from the stirring currents or arcs will lead to localized freezing.

(5) Sufficient space above the bath to accommodate any required electrode systems, both for general access and for adding electrodes, and free from falling chunks. It is possible that graphite electrodes could be introduced from the sides; plasma torches could more easily be side entry devices. Side entry will be essential for systems of direct transfer to melting.

(6) For a continuous melting unit, an iron tapping frequency of once every other hour and once each 6 to 8 hours for slag would be suitable for Plant 1.

4.1.2 Nature of the Material

(1) The sponge will be broken-off chunks, dropping from the reduction tubes. Their size and weight will be variable, but, in our opinion half-turn pieces will not be uncommon. If we assume that pitch equals diameter, we can approximate the weight of these chunks.

Tube Diameter [m]	Chunk Weight [kg]
0.30	5
0.45	17
0.60	40
0.90	150

Some of these weights pose problems. They are a threat to electrodes and will produce considerable splash.

(2) The sponge will contain melted or softened gangue - coal ash, wustite, and slag. Wherever fines can accumulate, a hanger or skull will develop.

(3) The sponge temperature will average 1100°C at tube discharge, with the helix surface hotter and the core cooler.

(4) The sponge will contain iron oxides and carbon. The latter can be sufficient to carburize the iron to any degree up to 4%.

(5) The sponge will melt down with gas evolution, inversely proportional to the degree of metallization. Most of the gas will evolve after the chunks are engulfed in slag, and will cause foaming.

(6) The sintered sponge is not "seriously" dusty; the inventors consider it to be not dusty.

4.2 Possible Transfer Systems

There are a number of possibilities for transferring material into the melting unit, each with its problems.

4.2.1 Unrestricted Gravity Transfer

This may not be operationally acceptable, because of the dimensions of the catchment zone and the impact of falling chunks. (It would be possible to make the melting unit long and narrow to accommodate the horizontal dispersion.)

4.2.2 Transfer by Chute

A chute would concentrate the material into a smaller area that could be better designed for melting. The chute would experience extreme conditions of impact, abrasion, atmosphere and temperature; a material to withstand this must be found. Also, viscous gangue in the material hitting the chute, will lead to skull build-up and accretion.

4.2.3 Transfer Through a Drag Conveyor Trough

A trough would give good focussing, but it is difficult to conceive of a drag conveyor operating successfully at 1100°C with large chunks.

4.2.4 Transfer by Collection in a Bucket

This would allow the melting unit to be a standard roof-charged steelmaking arc furnace. While the system required to accommodate and handle the buckets would be massive and mechanically complex, it could be made operable. This method would allow the reduction and melting units to operate semi-autonomously.

4.2.5 Cold Transfer

If transfer systems of an acceptable nature cannot be found, cold transfer could be developed. This is very unattractive because of the complexity and cost of cooling and the energy losses of the process.

4.3 Kinetics of Sponge Melting in a Bath

Heat transfer into the sponge may become a problem as tube system and chunk size increase. In the melting of DRI pellets in continuous charge practice, heat transfer takes place from slag to pellet almost exclusively. Pellets are immersed in the slag, slag is superheated in the arc zone, and strong fluid currents develop in the slag to carry heat to the pellets. In this practice, it is essential that pellets become immersed and dispersed in the slag; floating "islands" of pellets, once formed, are very slow to melt, and must be avoided.

The melting of chunks of sponge, weighing up to 150 kg will differ greatly from that of DRI. Fluid currents will not disperse heavy chunks, it is likely that the chunks will suppress the currents. The rate of heat transfer and melting of the solid will be much the same per unit of surface area in the chunk and in the DRI pellet. This implies a far longer bath residence time for chunks. The limiting weights of sponge particles charged to an electric furnace occurs in the melting of hot briquetted FIOR DRI. Individual briquettes might approach 0.3 - 0.5 kg. Extrapolating from this base to larger chunks requires study. Possibly, some computer modelling would help in assessing the magnitude of this problem, but experience may be the only sound guide to practice.

4.4 Batch Melting

One concept for dealing with the melting unit will see it as similar to electric steelmaking with scrap. In considering the problems of material transfer and the major unknowns involved in direct transfer and melting of chunks to a bath, batch melting appears to us to be the only practical route for Plant 1, and perhaps for Plant 2, without a massive development program.

Batch melting would proceed to steelmaking in the same furnace. The development problems besetting the LB process would be simplified. A proper question is, initiated as a batch melting process, whether it would then be the subject of a development program to convert to continuous melting. That would require both incentive and opportunity. A development program on direct melting would be much easier and economical, once an operating unit is producing chunky sponge. The problems associated with the chunks are central and major, and not readily dealt with on the usual pilot plant scale.

4.5 Continuous Feed Melting

Continuous melting can be envisaged in two different ways:

(1) Absorption directly into a bath. The earlier section on the kinetics of melting deals with this. The bath in this case would be open in initial concept.

(2) Melting out of a column of solid material superposed above a bath and continuously replenished. If this were possible, all problems of the heavy fall of chunks would disappear, as would some aspects of the wide distribution of the fall.

Melting out of a column would present its own problems:

(1) Electrical acceptability. A variable load of conductive sponge bearing on the electrodes could perhaps be dealt with through responsive electrical control systems.

(2) Cold taps. The column of unmelted material would force its lower levels down into the slag and metal, and fluid currents in these layers would be suppressed. The temperatures at tap would be determined mostly through the limitations imposed by these two conditions.

(3) Hangups. Hangup of the column would be disastrous, as it would lead to column collapse with mechanical stresses and violent gas release in the furnace system.

The possibility exists of melting out of a continuously-replenished column of solid material could be combined with steelmaking. Doing this, however, would require interruption of the sponge stream for periods of at least 15 minutes every hour or so, a condition very difficult to accept in the reduction unit and probably impractical.

5.0 PROBLEM AREAS

We have visualized some aspects of how a commercial plant based on the LB Process might be arranged. In doing so, we have made assumptions and brushed aside major problem areas. The commercial visualization that has resulted is probably a best case. In this section, we deal with some of the problem areas. It is our expectation that some will be resolvable by analysis or small scale trial. Others may not be resolvable except through large scale trial.

1) Heat transfer inside the tubes. The productivity assumption is defined entirely in terms of outer tube surface area and is insensitive to tube diameter. This results from the assumption that there is no resistance to heat transfer inside the tube. That is patently insupportable. The tube wall will thicken as diameter increases. Even in the laboratory, the sponge discharged from the tube showed gradients from surface to central core - evidence of an internal resistance to heat transfer. Putting internal resistances to heat transfer into the analysis of productivity is complex. The process inventors, we understand, are doing some studies on the subject and we hope will have some results to consider before our work is finished. If not, we intend to do sufficient to have an appreciation of its impact.

2) Dealing with large chunks. Scale-up to commercial plants will result in an increased problem in dealing with large chunks of sponge iron. We have assumed that they can be managed physically, and, in effect, that the problem is soluble through design. This is probably true in regard to straight material handling, so long

as the sponge can be counted on to break off in manageable lengths. Trial on the large scale will be essential for melting sponge chunks and can probably be done on the pilot scale by accumulating sufficient sponge to allow melting trials in a full-size steelmaking furnace.

3) Structural aspects of screws, tubes, and supports. We expect to carry out some analysis and preliminary design studies in these areas. We understand that silicon carbide tubes have been made to 0.45 m diameter and to about 1 m length. The tubes will be under tensile stress from the screw action, and bending stresses from cantilevered loads. The problems encountered when refractories are in tension are well known.

4) Combustion. We are assuming that the gases produced in the reduction unit will be suitable for combustion without cleaning. This is a part of the rationale for producing chunky material. The Lakeview pilot plant should aim to verify this.

5) Shutdown and interruption performance. The performance of the reduction unit through interruption cycles will need some study, taking into consideration the large thermal flywheel in the refractory enclosure. Drive failure on a screw, and the consequences on restarting of the rise to furnace temperature of the tube, screw and contents, need study.

(6) It has been assumed that the production of iron from fine unagglommerated constituents is generally desirable economically. However fine materials have their drawbacks. The technology for handling fine coal safely is known. The transport, bulk storage and mixing problems when fine iron ore particles are transported in wet, windy and freezing conditions will require study.

Mixing and delivery of fine ore-coal mixtures to forty feeding inlets is a major materials transport problem for the multi-tube reactor concept.

6.0 PROGRAM OF TECHNICAL EVALUATION

Working with the commercial-scale models proposed in this report, we intend to bring the following analyses and studies to bear:

1) For each section (reduction-combustion, transfer, and melting-steelmaking), heat balance and energy demand calculations. This will include examination of how best to manage combustion within proper temperature limits.

2) A diagram setting out the flowsheet - material balance relationships.

3) For the screws and tubes, some aspects of their structural requirements.

4) An approximation to internal resistances to heat transfer and their significance.

5) An examination of the internal temperature history on failure of a screw drive.

6) Development of steelmaking practice

7) Evaluation of operability (start-up, shutdown, maintenance, interruptions, etc.).

7.0 REFERENCES

Paper 3 The LB Furnace for Smelting Reduction of Iron Ores. W.K. Lu, C. Bryk and H.Y. Gou, Process Technology Proceedings, Vol. 6, (Fifth International Iron and Steel Conference, Book), Washington, D.C., 1986, Iron and Steel Society Inc.

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HATCH
ASSOCIATES

INTERNAL MEMO
I.M. #5006.14
File #5006.3

DATE: December 14, 1987

TO:
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FROM: J. Jones

LB PROCESS EVALUATION

Vertical Configurations

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APPENDICES

1.0 INTRODUCTION

Due to mechanical and structural problems with the proposed commercial LB unit (see report 5006.1 Rev.1), the inventors proposed an alternate vertical configuration (see Fig. 1). This consisted of two concentric silicon carbide tubes (4.0 m ID and 4.6 m ID) with the ore/coal mixture flowing in the annulus between the tubes. It was proposed that CO gas would be combusted and recycled along both sides of the annulus in order to improve heat transfer. Additional heat would be recovered by preheating air in a tube extending down inside the inner SiC tube. Iron would be discharged from the shaft annulus and would fall into a packed column. Hot gases and plasma torches would be used to melt the iron out of this column whereupon it would flow into a bath. Iron could then be tapped periodically from the bath.

In order to split the CO flow from the annulus it would be necessary to have perforations in the outer SiC tube at its base. This would allow gas to escape into the chamber surrounding the outer SiC tube.

The ore/coal mixture would be fed by a large helical ribbon which would fit inside the annulus. The helix is to be supported from above by the drive system. The helix pitch length would be 15-20 cm, to ensure that the size of sponge iron chunks exiting the annulus will be small (~ 30 kg).

Much higher operating temperatures have been proposed (as high as 1500°C on the SiC tube wall at the bottom of the reaction chamber). It is felt by the inventors that these higher tempertures will give much greater heat transfer and that reaction rates will increase significantly. The ore/coal bed thickness is 30 cm, but since heat transfer is from both sides of the bed the distance that heat must be transferred through is only 15 cm.

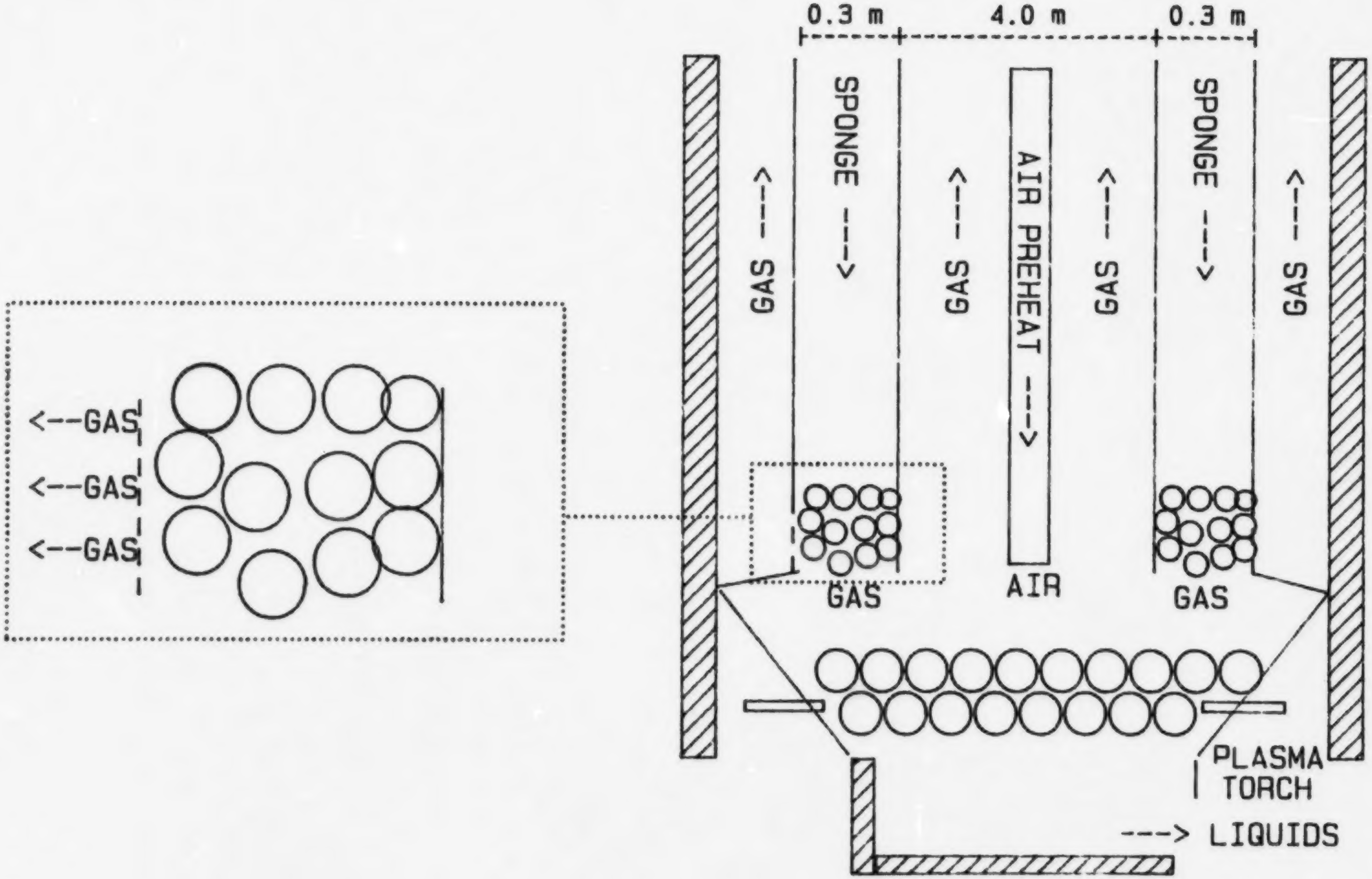

0.3 m
4.0 m
0.3 m
GAS --->
SPONGE --->
GAS --->
AIR PREHEAT --->
GAS --->
SPONGE --->
GAS --->
GAS
AIR
GAS
PLASMA TORCH
---> LIQUIDS
<--GAS
<--GAS
<--GAS

Fig.1 LB Furnace Configuration
Vertical 300,000 t/y Unit

2.0 HEAT TRANSFER/REACTION RATES

Heat transfer has been one of the major concerns for the LB process. By adopting a design with material flow within an annulus it is possible to have two sided heat transfer. The surface area available for heat transfer is greatly increased without increasing the overall size of the reaction unit.

The proposed operating temperature of 1500°C for the tube wall at the bottom of the reactor will give greater heat transfer through the reactor wall and should also lead to greater reaction rates, though there is no experimental data to support this. Potentially higher temperatures could lead to localized melting of material adjacent to the reactor walls. This might result in:

a) Sticky material adhering either to the screw or to the walls of the annulus, resulting in jamming.

b) Deterioration of the SiC tube. At these temperatures SiC may dissolve in the molten ferrous material. Suppliers have indicated that erosion of the SiC, because of abrasion, might be a problem at these temperatures.

These temperatures would make it impossible to extend the helix conveyor the full length of the annulus unless it was made of a heat resistant material. The inventors have suggested that the lower section be made of SiC, but we believe that the helix would not last because of the torsion that it would be subjected to. In addition, the differential between steel and ceramic thermal expansions make it very difficult to join the two materials.

The inventors prefer that the material exiting the annulus be 80% reduced. The remaining reduction would take place while material lays in the packed column above the bath. In this case, the material can take advantage of radiative heat transfer from the combustion gases before they flow up along the reactor walls. One major concern here is whether or not the material in the packed column will be exposed to an oxidizing atmosphere. If so, the material will rapidly reoxidize at these high temperatures. One way to prevent this is to ensure that there is sufficient excess carbon present to prevent reoxidation, as practiced in kiln processes. However, this will result in an increase in the energy requirement since coal is the major source of process energy. Calculations by the inventors (see Appendix A) indicate quite high off-gas temperatures from the unit (~1100°C) for 80% reduction. Some of this heat could be recovered by preheating the combustion air but there does not seem to be any way to recover a major portion.

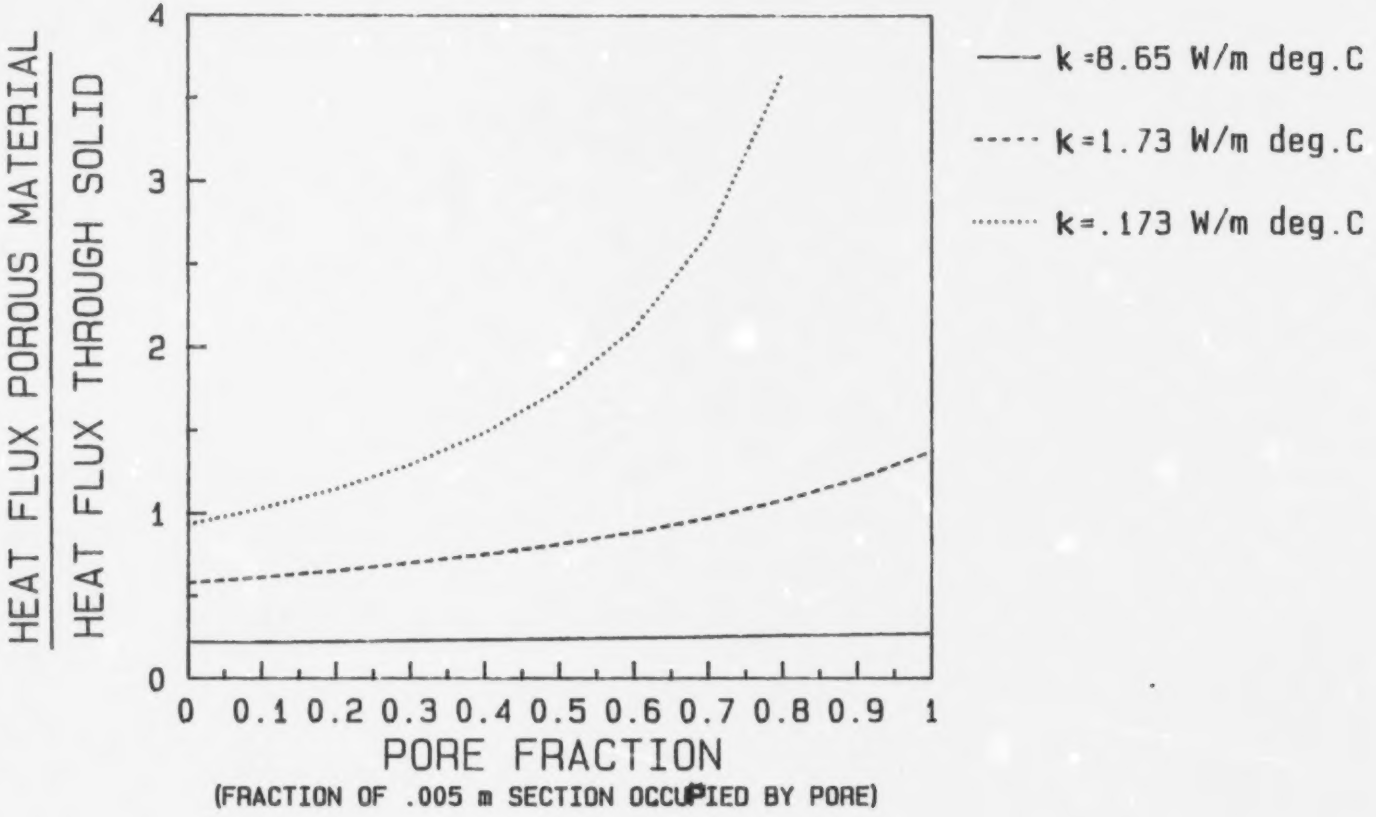
Reaction rate data has been obtained in muffle furnace experiments performed between October 26, 1987 and November 17, 1987. Centreline temperatures in loaded crucibles and penetration of the reaction front into the ore/coal mixture were used to determine effective reaction rates for the observation period (see Appendix B). Unfortunately, some of the results were inconsistant. One factor causing this was the packing density, as was the case in two of the runs on November 17th when there was a gap between the ore/coal mixture and the tube wall. This resulted in the centreline temperature increasing up to 5 times more rapidly than had been obtained in previous experiments as the thermocouple was receiving additional radiation. From these preliminary tests, effective reaction rates for the metallization front ranging from 0.2 mm/min to 1.04 mm/min have been obtained. For the higher rates the required time to react a bed 30 cm thick would be on the order of 17.5 hours, assuming a linear progression of the front. With an annular arrangement the bed thickness would be 15 cm and the required residence time in the reactor would be 8.75 hours. (It should be noted that these higher reaction rates were obtained for cases where there was direct radiation to an exposed face of the reacting material at muffle furnace temperatures of 1000°C - 1400°C over the reaction period.) However, at the higher desired operating temperatures of 1500°C plus for the commercial unit, it is not possible to determine effective reaction rates until sufficient experimental work has been carried out. The inventors have estimated that the screw conveyor would heat up rapidly and would provide internal heat transfer to the ore/coal mix. To what degree this would help shorten the required residence time in the reactor cannot be determined until more experimental work is performed. (One method of determining the effect of the screw would be to place a section of screw in a tube and pack it with an ore/coal mix. The reaction time for this case could then be compared to the case where no screw is used.) The inventors tried a test with a plate of steel in the middle of the crucible and the effective reaction rate was increased 2.5 times, with the plate exposed to direct radiation.

It has been proposed by the inventors that because of the high temperatures at which the material was being reacted, the primary mode of heat transfer would be by radiation across the pores of the reduced sponge. The inventors suggested that the effective thermal conductivity of the material would be several times greater than that of solid iron and therefore, internal resistance to heat transfer would not be a determining factor in limiting the reaction rate.

In order to estimate this effect a computer model was constructed to evaluate the ratio of heat flux through porous material to that through solid material for various emmisivities and thermal conductivities (see Appendix C). The model indicated that only for very low thermal conductivities ($\sim$ 1.7 W/m°C) and high porosities ($\sim$ 0.8) would the heat flux through porous material be greater than or equal to the heat flux through solid material (see Fig. 2). For sponge iron the porosity could be expected to be 0.5. A thermal conductivity is given for one sponge iron (Bogdandy p. 120) as being ~ 4.7W/M°C. Our model would indicate that the sponge iron would inhibit heat transfer to the unreacted material and would affect the reaction rate.

One item requiring further definition is that the effective reaction rates have been obtained for stationary beds exposed to muffle furnace temperatures of 1000°C - 1400°C over the observation period. In the commercial unit less than half the unit would be exposed to the higher temperatures and in most of the unit the heat transfer will be by conduction. For these preliminary tests, there was heat transfer by radiation to the exposed face of the reacting material. This will only occur at the exit of the reduction unit. The test results will give optimistic values for the effective reaction rate. Therefore the residence times that could be expected if the operating temperatures were in the range of 1000°C to 1400°C must be examined. It is not possible to comment on the effective reaction rates and required residence time for an operating temperature of 1500°C.

FIG.2 HEAT FLUX THROUGH SOLID/POROUS MATERIAL VS PORE SIZE FOR VARIOUS THERMAL CONDUCTIVITIES



3.0 MECHANICAL CONSIDERATIONS

The major mechanical concern for the vertical LB unit is the helical feeder. The helix can be supported at the top where it is attached to the drive mechanism, but it does not seem possible to support the helix at the bottom. The operating temperature at the reactor bottom will be too high for a support mechanism for the helix unless it is water-cooled and this would impede the flow of sponge out of the bottom of the annulus. If the helix is supported only from the top it becomes an infinite length ribbon with no structural stability.

It is not possible to use steel at the hot end of the tube and it may be necessary to use some other material for the screw in this portion of reactor. In conversations held with Carborundum, the senior field engineer stated that he saw no way of attaching a SiC helix tip to a steel helix because of the different thermal expansions of steel and SiC. The torque on the helix may be too great to use a ceramic even for the tip of the helix.

As an alternate, the inventors have suggested that refractory helix flights be attached to the inner wall of the annulus and that this structure be rotated. Beyond the technical difficulty of attaching the refractory, torque would be placed on the flights causing them to shear in tension. This does not appear to be a feasible solution to the helix construction problem.

It is desired that the helix pitch be kept small (~ 15-20 cm) in order to limit the size of the sponge iron chunks. This means that the length of the helix will be very great.

Several difficulties must be overcome regarding the structure of the helix, the material of construction and the method of support. Gravity feed of material may not be a feasible solution either because of sticking of materials.

4.0 STRUCTURAL CONSIDERATIONS

The primary structural considerations for the vertical LB unit regard the materials of construction because of the high operating temperatures and the gases present in the system. As mentioned previously, it is likely that molten material will be present if temperatures as high as 1500°C are used. This will result in loss of SiC from the tube walls as it will dissolve in the molten material. Carborundum representatives indicated that temperature is usually the critical factor that affects the wear and life of silicon carbide (see Appendix D).

The gases that will come into contact with the silicon carbide tubes can react. Our mass balances (Appendix B) indicate that the reaction gas produced from the iron oxide reduction will consist mainly of CO, H_2 and approximately 3% H_2O. Information supplied by Carborundum indicates that the following chemical reactions will result. Hydrogen will give moderate attack at 800°C and rapid attack at 1100°C. Steam will result in oxidation of silicon carbide in the range of 800°C - 1200°C, with the greatest attack occurring at 1000°C. (In this temperature range, Carborundum recommended that a specially coated silicon carbide be used.) This coated material would not withstand temperatures above 1100°C-1200°C. Carbon monoxide leads to carbon deposits below 580°C and slow attack of the silicon carbide via the reaction $SiC + 2CO \rightarrow SiO_2 + 3C$ at 1200°C. Carbon dioxide leads to slow attack via the reaction $SiC + 3CO_2 \rightarrow SiO_2 + 4CO$ at ~ 800°C. The reaction of major concern is oxidation of silicon carbide by water vapour. Once the reaction gas is combusted, the mole fraction of water increases to 10%.

If the coated silicon carbide is used, this requires that the temperature profile along the full length of the reactor be known to determine which section must be made of the coated silicon carbide. If fluctuations occur in off-gas temperatures, or at start-up and shut-down problems will arise because of reactions outside of the desired temperature range.

Of secondary concern is the thermal shock that the silicon carbide will experience. The initial unit was to operate continuously for five days with two days for maintenance and repair. If this were to be the case, Carborundum representatives indicated that the life of the silicon carbide would be limited. Shutdowns once a month greatly decrease the life of the silicon carbide. In previous applications it was found that frequent heating and cooling led to rapid deterioration of silicon carbide while continuous operations gave a silicon carbide life of up to 10 years.

Another concern is the structural support for the annulus. For smaller diameters (up to 1.3 m) silicon carbide rings can be joined together with silicon carbide cement. The structural characteristics of this bond must be determined. The rings could be supported with a T-bar type column, though this will only provide support for compression and would not resist tension which will occur from the screw action and radial pressure put on the annulus walls when conveying the material down the annulus. For the larger diameters (4.0 - 4.3 m) required for the commercial unit, we believe it would be necessary to use bricks.

If bricking were used, additional support could be provided by support beams attached to the outer wall of the unit. Whether or not these beams would severely affect the flow of gas remains to be seen. The thickness of the bricks would result in additional resistance to heat flow and would require that the residence time in the reactor be increased.

Support of the inner wall could possibly be with some sort of arched construction to the discharge shaft/chute. The material could fall into the shaft though the open arches if the reduced broken-off pieces were not too long. Uniformity of burning of the exit carbon monoxide with the preheated air and distribution between the inner and outer walls is a complex problem which must be developed.

In closing, the major structural questions concern the materials of construction to be used for the reactor tubes. It is not readily apparent that silicon carbide will be suitable. If another high temperature ceramic is to be used, heat transfer would be adversely affected if the material used does not have a thermal conductivity similar to or better than silicon carbide. The method of structural support for the annular arrangement also poses problems because of the forces that will be exerted outwards on the annulus walls by the material flowing in the annulus and by the screw feeder itself.

5.0 DESIGN PROBLEMS

In any commercial plant operating problems occur. However, certain of these forseeable operation problems stand-out as technical problems that should be resolved before any further attempt is made to produce a commercial design. Some of these problems have been discussed in previous sections of this report (e.g effect of thermal shock on life of silicon carbide). Others will be discussed here.

One of the major concerns from an operability point of view is material transfer. Potential problems regarding the screw feeder have been discussed. Other problems include gas flow, packing density, and material flow at the exit of the annulus.

Packing in an annulus requires that the pitch decrease along the axis. The inventors have stated that they wish to pack the ore/coal mixture as tightly as possible in order to increase throughput. This poses several problems. The mix can not be packed too tightly if the screw feeder is unsupported, because the flights will not be able to support the load. It might be possible to make the flights thicker, but this will add to the weight of the screw feeder which is not desirable. Another problem is the fact that at a temperature of 800°C, coal swells approximately 20-30% before it starts to react via the Boudourd reaction. As a result, it would probably be necessary to fill the flights only to a level of about 80% of capacity in order to allow for expansion of the material. Otherwise additional support would be necessary in order to take the additional forces on the walls. Swelling characteristics and packing densities must be established for specific ore-coal mixtures.

Below the annulus exit, the reacted material will sit in a packed column and is to melt out of the column into the bath. During the period that the material is sitting at the top of the bed it will be exposed to an atmosphere consisting most of CO_2 and H_2O (since combustion of the reaction gases takes place above the bed). As a result, it is likely that some of the reacted material will re-oxidize.

Solid material transfer also has several problems associated with it. One of the major concerns is the flow of fine materials out of the annulus. This fine material will come from two sources. The first of these is fine material that flows down the annulus walls. This may be kept to a minimum by having a snug fit between the screw conveyor and the annulus walls, but allowance must be made for the thermal expansion of the helix at high temperatures. The second source of dust will be from the reacted material at the annulus exit. The inventors wish to have the material exiting the annulus 80% reduced. As a result, 20% of the material exiting will be unreacted, likely in the form of an unreacted core. As the sponge iron breaks up the unreacted powder will be released above the packed bed. This will result in dirty off-gas and will retard heat transfer.

Even with a 15-20 cm pitch, it is likely that the sponge chunks breaking off at the bottom of the annulus will be large due to the large diameter (4.0 m ID, 4.6 m O.D.) of the annular system. If the chunks are large then it will take considerably longer to melt the material out of the packed bed into the bath.

The mechanics of sponge flow down the shaft and melting at the bottom of the shaft merit study. If the material breaks off as elongated jackstraws, movement down the shaft of the hot sticky material might be hindered. Also shaft melting of this material may be more akin to cupola melting, wherein material flow and melting is dependent on small regularly sized material with a coke bed in the shaft to aid the process. This part of the process may be difficult to assess at bench-scale, as stated in our first phase report, but could be left aside until many of the other experimental questions are examined.

6.0 CLOSING COMMENTS

This study was commissioned to deal with the question of whether there is justification to initiate pilot plant study on a rather large scale. A positive answer to this question assumes that a promising commercial plant can be visualized and the small scale work has dealt systematically with the basic reaction mechanisms and concepts. Satisfying the first of these is not possible if the second is not satisfied. We have been unable to visualize an acceptable commercial plant, because there are many areas where no, or only sketchy, data is available.

In a sense, we have "invented" problems, which we consider to be of substance, and to some of which we think answers should arise from small-scale or laboratory work. Here is a listing of the latter:

- Heat transfer at elevated temperatures (1500°C)
- Reaction rates at 1500°C
- Swelling characteristics of specific coal-ore mixtures for typical thermal cycles
- Reactivity of silicon carbon to liquid iron - iron oxide mixtures at upper temperature range
- Reactivity of silicon carbide to process gases over the full temperature range
- Determination of amount of excess carbon required to prevent oxidation of the sponge mass when exposed to the preheated air and to the combustion gas products for times approaching those in any commercial unit
- Establish tests to determine if a helical screw buried in an ore coal mixture contributes to heat transfer
- Establish test procedure to determine rate of progress of reaction front as a function of distance at a constant heat flux. This will probably follow some practical power law rather than the linear approximation we have used
  (Distance = $(\text{time})^{0.4 - 0.5}$)

APPENDIX A Mass and Energy Balance for the LB Process

MASS AND HEAT BALANCE AND HEAT TRANSFER
IN LB FURNACE PROCESS

HARRY GOU Nov. 18, 1987

The overall Heat and mass balances in LB Process based on 1000 kg Hot metal Production have been made. The whole process then divided into two zones, reduction zone and melting zone. The staged heat and mass balances in these two zones are calculated, respectively, according to diffesent spong metallization degree η. The η value has no influence on overall mass balance, but it affects the electric energy requirement, so affects the overall heat balance.

The heat transfer in one-dimension cylindrical metal layer is caculated, represented by reduced metal layer thickness. The influence of effective heat conductivity k_e on the reduced layer thickness Δr shows a non-linear relationship between k_e and Δr.

MASS BALANCE

1. RAW MATERIALS COMPOSITION:

IRON ORE CONCENTRATES:

ORE	Fe_3O_4	SiO_2	Al_2O_3	CaO	MgO	MnO	TiO_2	Na_2O	K_2O	P_2O_5	S
w. %	93.01	5.16	0.58	0.28	0.35	0.014	0.02	0.006	0.06	0.03	0.11

COAL COMPOSITION:

COAL	C_{Fix}	V.M.	S	Ash	Σ
w. %	62.76	31.25	0.73	5.76	100.

2. THE MIXTURE ORE / COAL RATIO.

ORE / COAL = 80 / 20.

3. HOT METAL AND SLAG COMPOSITION: TOTAL RECOVERY IS 99.7%

HM	Fe	C	Si	Mn	P	S	Σ
w. %	98.69	1.0	0.25	0.01	0.02	0.03	100.

HOT METAL

SLAG	FeO	SiO_2	Al_2O_3	CaO	MgO	MnO	TiO_2	Na_2O	K_2O	P_2O_5	S	Σ
w. %	2.0	35.94	6.28	35.96	13.77	0.03	0.13	0.30	0.35	0.04	1.15	100.

SLAG.

4. EXT GAS COMPOSITION: COMPLETE COMBUSTION. EXCESS AIR IS 8%.

EX. GAS	CO_2	CO	N_2	H_2O	O_2	Σ	
V. %	23.18	0.0	64.97	10.90	1.35	100.	

5. DISCUSS:

The sponge Metallization degree does not affect the overall mass balance. Its effect appear in "stages mass balance" between Reduction zone and melting zone.

HEAT BALANCE

1. COAL HEAT:

COAL CALORICITY: 7900 KCAL / kg Coal.

2. HEAT CAPACITY OF COMPOSITE:

$$C_{p_i} = a_i + b_i T + c_i T^{-2} \quad , \quad C_{p_\Sigma} = \Sigma C_{p_i}$$

3. COMBUSTION.

COMPLETE COMBUSTION. EXCESS AIR, NO CO OR H_2 IN EXIT GAS.

4. HEAT LOSS.

10% OF HEAT SUPPLIED ACCOUNT FOR HEAT LOSS THROUGH WALL.

5. HOT METAL AND SLAG TEMPERATURE.

HOT METAL AND SLAG TEMPERATURE IS 1500°C.

6. DISCUSS.

The 80/20 ORE/COAL ratio is close to stoichiometric requirement of direct reduction for iron oxides. With the amount of coal the heat is already enough for whole process unless the heat in exit gas can be recovered. Changing the metallization degree of sponge influence the electric energy requirement. Less metallized sponge required more heat in melting zone, and increases the exit gas temperature and its sensible heat.

OVERALL MASS BALANCE

INPUT	Kg	OUTPUT	Kg
IRON ORE	1465	HOT METAL	1000
COAL	366	SLAG	223
CAO	85	EXT GAS	3283
MgO	25		
AIR	2565		
TOTAL	4506	TOTAL	4506

OVERALL HEAT BALANCE (η=99.7)

SUPPLIED	GCAL	%	REQUIRED	GCAL	%
COAL HEAT	2.9	91.5	REACTION HEAT	1.58	49.8
ELEC. HEAT	0.27	8.5	EX. GAS SENSIBLE HEAT	0.81	25.6
			HM AND SLAG SENSIBLE HEAT	0.44	13.9
			HEAT LOSS	0.34	10.7
TOTAL	3.17	100	TOTAL	3.17	100.

STAGED MASS BALANCE (kg/THM)

METALLIZATION OF SPONGE

I. REDUCTION — II. MELTING

η=98.7

INPUT	kg	OUTPUT	kg	INPUT	kg	OUTPUT	kg
IRON ORE	1465	SPONGE	1228	SPONGE	1228	HOT METAL	1000
COAL	366	EXIT GAS	3283			SLAG	223
CaO + MgO	110					GAS (M)	5
AIR	2565						
GAS (M)	5						
TOTAL	4511	TOTAL	4511	TOTAL	1228	TOTAL	1228

η=80

INPUT	kg	OUTPUT	kg	INPUT	kg	OUTPUT	kg
IRON ORE	1465	SPONGE	1326	SPONGE	1326	HOT METAL	1000
COAL	366	EXIT GAS	3283			SLAG	223
CaO + MgO	110					GAS (M)	103
AIR	2565						
GAS (M)	103						
TOTAL	4609	TOTAL	4609	TOTAL	1326	TOTAL	1326

η=95

INPUT	kg	OUTPUT	kg	INPUT	kg	OUTPUT	kg
IRON ORE	1465	SPONGE	1252	SPONGE	1252	HOT METAL	1000
COAL	366	EXIT GAS	3283			SLAG	223
CaO + MgO	110					GAS (M)	29
AIR	2565						
GAS (M)	29						
TOTAL	4535	TOTAL	4535	TOTAL	1252	TOTAL	1252

* GAS (M) — gas from melting zone entering Reduction zone.

STAGED HEAT BALANCE (GCal/THM)

$\eta = 99.7$

I. REDUCTION

SUPPLIED	GCAL	%	REQUIRED	GCAL	%
COAL HEAT	2.9	100	REACTION HEAT	1.58	54.0
GAS(M) SENSIBLE HEAT	~0	0	SENSIBLE HEAT EXIT GAS	0.81	28.0
			SPONGE SENSIBLE HEAT	0.22	8.0
			HEAT LOSS	0.29	10.0
TOTAL	2.9	100	TOTAL	2.9	100

II MELTING

SUPPLIED	GCAL	%	REQUIRED	GCAL	%
SENSIBLE HEAT SPONGE	0.22	45.0	HM SENSIBLE HEAT	0.33	67.0
ELEC. HEAT	0.27	55.0	SLAG SENSIBLE HEAT	0.11	22.0
			GAS(M) SENSIBLE HEAT	~0	0
			HEAT LOSS	0.05	11.0
TOTAL	0.49	100	TOTAL	0.49	100

TOTAL HEAT INPUT = 3.17 GCAL, TOTAL HEAT REQUIRED = 3.17 GCal

$\eta = 80$

SUPPLIED	GCAL	%	REQUIRED	GCAL	%
COAL HEAT	2.9	98.6	REACTION HEAT	1.26	42.8
GAS (M) SENSIBLE HEAT	0.04	1.4	SENSIBLE HEAT EXIT GAS	1.15	39.1
			SPONGE SENSIBLE HEAT	0.24	8.2
			HEAT LOSS	0.29	9.9
TOTAL	2.94	100	TOTAL	2.94	100

SUPPLIED	GCAL	%	REQUIRED	GCAL	%
SPONGE SENSIBLE HEAT	0.24	27.3	HM SENSIBLE HEAT	0.33	37.5
ELEC. HEAT	0.64	72.7	SLAG SENSIBLE HEAT	0.11	12.5
			GAS(M) SENSIBLE HEAT	0.04	4.5
			REACTION HEAT	0.32	36.4
			HEAT LOSS	0.08	9.1
TOTAL	0.88	100	TOTAL	0.88	100

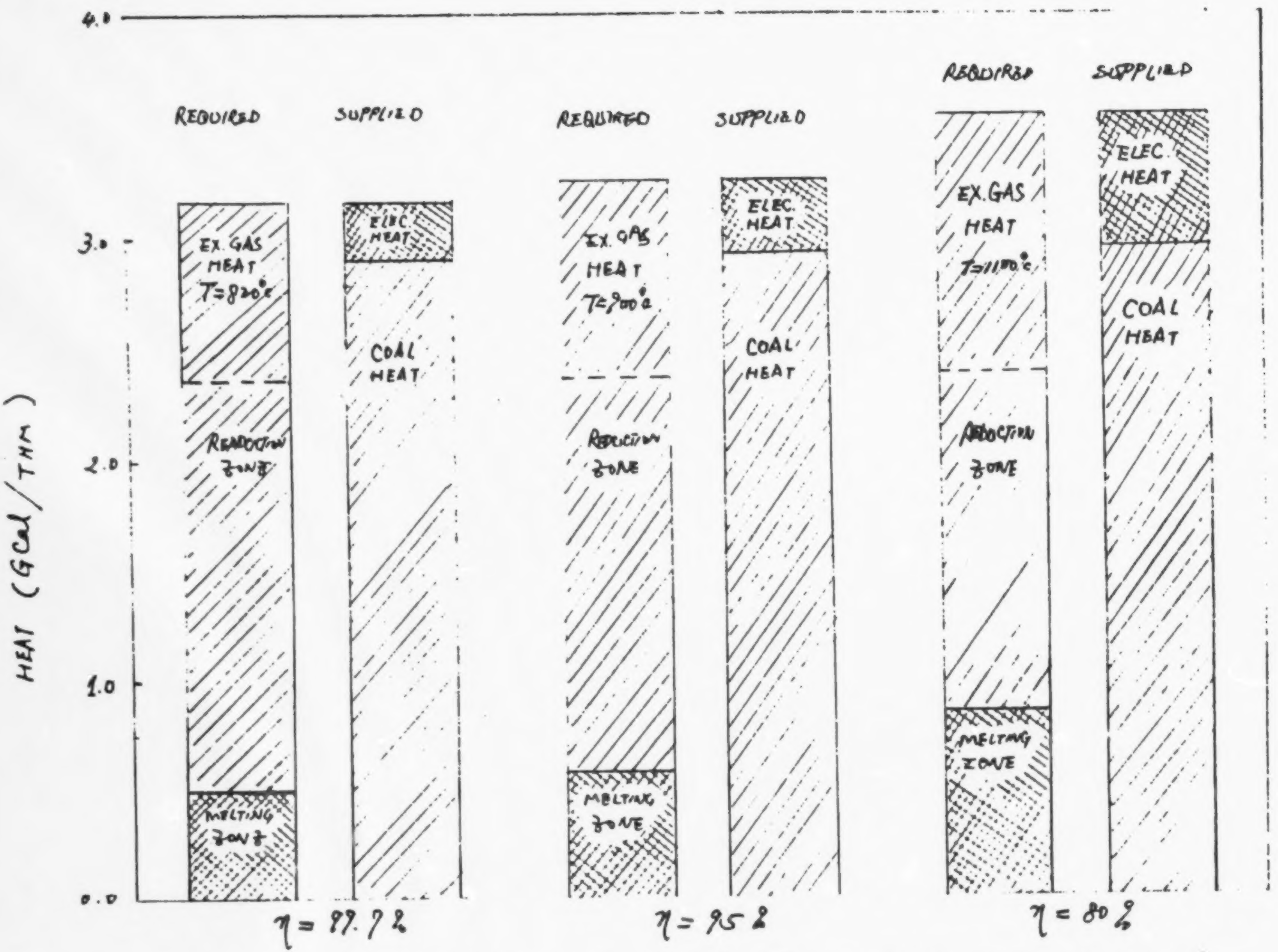
TOTAL HEAT INPUT = 3.54 GCAL

$\eta = 95$

SUPPLIED	GCAL	%	REQUIRED	GCAL	%
COAL HEAT	2.9	99.7	REACTION HEAT	1.50	51.5
GAS (M) SENSIBLE HEAT	0.01	0.3	EXIT GAS SENSIBLE HEAT	0.89	30.6
			SPONGE SENSIBLE HEAT	0.23	7.9
			HEAT LOSS	0.29	10.0
TOTAL	2.91	100	TOTAL	2.91	100

SUPPLIED	GCAL	%	REQUIRED	GCAL	%
SPONGE SENSIBLE HEAT	0.23	39.7	HM SENSIBLE HEAT	0.33	56.9
ELEC. HEAT	0.35	60.3	SLAG SENSIBLE HEAT	0.11	19.0
			GAS(M) SENSIBLE HEAT	0.01	1.7
			REACTION HEAT	0.08	13.8
			HEAT LOSS	0.05	8.6
TOTAL	0.58		TOTAL	0.58	100

TOTAL HEAT INPUT = 3.25 GCAL



η – SPONGE METALLIZATION DEGREE: HEAT REQUIRED INCLUDING 10% OF HEAT SUPPLIED AS HEAT LOSS

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 1 OF			
DESCRIPTION Flowsheet Material Balance	PROJECT NO.	MADE BY AGWL okGWL DATE Oct 6/87	CHECKED BY JATJ DATE Oct 14/8

1) Quantities of ore and melting slag.

Based on 1000 units hot metal
Losses of iron to dust etc = 2%

iron in hot metal = 98.7 wt
iron in slag = 2 wt%
iron in ore = 68 wt%

Let a = # units of ore , b = # units of slag produced

Iron balance: $0.68\,a = 1000 \times 0.987 + 20 + 0.02\,b$
$= 1007 + 0.02\,b$

SiO_2 balance: (assume coal ash = 65% SiO_2 ;
Based on 350 units of coal

ash in coal = 4 wt
SiO_2 in ore = 5 wt
SiO_2 in hot metal = 0.25
SiO_2 in slag = 36 wt

$0.65 \times 0.04 \times 350 + 0.05\,a = 1000 \times 0.0025 + 0.36\,b$
$0.05\,a = -7.0 + 0.36\,b$

Solving the two mass balance equations
we find that $a = 1502$
$b = 228$

2) Reduction gases (Based on 350 units coal)

Amount of entering moisture = 350 × 3% = 10.5 units
Amount of entering volatiles = 350 × 32% = 112 units

From Lu's paper

volatiles composition
C/H+O/N = 19/9.6/1.

and H/O = 4.9/4.7

∴ Amount of C in volatiles $= \frac{19}{30.1} \times 112 = 70.7$ units
Amount of fixed C in coal $= 0.63 \times 350 = \underline{220.5}$ units
total C = 291.2

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 2 OF

DESCRIPTION	PROJECT NO.	MADE BY DATE	CHECKED BY DATE

Amount of N_2 : $1.5/30.1 \times 112$ = 5.6 units

Amount of H+O in entering volatiles = $\frac{9.6}{30.1} \times 112 = 35.7$

Amount of H in entering volatiles = $\frac{4.9}{9.6} \times 35.7 = 18.2$

Amount of O in entering volatiles = $\frac{4.7}{9.6} \times 35.7 = 17.5$

∴ Amount of H total in coal : in water $\frac{2}{18} \times 3\% \times 350 =$ 1.2
in volatiles = 18.2
total 19.4

Amount of O total in Coal : in water $\frac{16}{18} \times 3\% \times 350 =$ 9.3
in volatiles = 17.5
total 26.8

Amount of C total in coal : fixed carbon 220.5
in volatiles 70.7
total 291.2

Oxygen removed in reduction unit :

Entering with iron (magnetite) : $\frac{64}{168} \times 1502 \times .68$ = 389 units Oxygen

Iron entering = 0.68×1502 = 1021.4 units

Iron metallized (90%) × 1021.4 = 919.3 units

Iron left as FeO = 102.1 units

Oxygen in FeO = $\frac{16}{56} \times 102.1$ = 29.2 units.

∴ Oxygen removed = 360 units .

Carbon Required in melting unit:

Oxygen in entering FeO = 29.2 units

Fe left in slag as FeO = 0.02×228 = 4.6 units

Fe lost as dust (2%) = 20 units

total Fe lost 24.6 units

Oxygen in lost Fe = $\frac{16}{56} \times 24.6$ = 7.0 units

Oxygen removed in melting unit = 22.2 units

Amount of C needed = $\frac{12}{16} \times 22.2$ = 16.6 units .

Carbon in Hot metal 1% × 1000 = 10 units

∴ total C in feed to melter = 26.6 units

Carbon in reduction unit offgases :

Carbon added = 291.2 units

Carbon in melterfeed = 26.6 units

∴ Carbon in reduction gases = 264.6 units (22.05 moles) .

Hydrogen in reduction unit offgases :

From previous calculation H_2 in reduction gases = 19.4 units (9.7 mols)

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 4 OF

DESCRIPTION	PROJECT NO.	MADE BY DATE	CHECKED BY DATE

Oxygen in reduction unit off gases:

From ore = 360 units
From coal = 26.7 units

total 386.7 units (24.17 mols)

Reduction Off-gas Composition: H_2O/H will be ~ double CO_2/CO.

C/H_2 ratio in gas: weight $264.6 / 19.4$

Volume $\frac{264.6}{12} / \frac{19.4}{2} = 2.273$

$$V\%(CO+CO_2) / V\%(H_2+H_2O) = 2.273$$

and $V\%(CO+CO_2) + V\%(H_2+H_2O) = 100$

$$\therefore V\%(CO+CO_2) = 69.4$$
$$V\%(H_2+H_2O) = 30.6$$

In mols

$CO + 2CO_2 + H_2O = 24.17$
$CO + CO_2 = 22.05$
$H_2 + H_2O = 9.7$
total = 31.75.

$$\therefore CO_2 + H_2 = 2.12$$

$H_2O/H_2 = 2CO_2/CO \quad \therefore H_2 = H_2O \times CO/2CO_2$

$$H_2O \times \frac{CO}{2CO_2} + H_2O = 9.7$$

$$H_2O(1 + CO/2CO_2) = 9.7$$

$$H_2O\left(\frac{2CO_2 + CO}{2CO_2}\right) = 9.7$$

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 5 OF

DESCRIPTION	PROJECT NO.	MADE BY DATE	CHECKED BY DATE

$H_2O = 19.4\,CO_2 / (2CO_2 + CO)$

but $CO_2 + H_2O = 2.12$

$\therefore \quad CO_2 + 19.4\,CO_2 / (2CO_2 + CO) = 2.12$

$CO_2 + 19.4\,CO_2 / (CO_2 + 22.05) = 2.12$

solving for CO_2 we find that $CO_2 = 1.15$

$\therefore \quad CO = 20.9$ and $CO_2 / CO \simeq 0.05$

$H_2O = 2.12 - 1.15 = 1.0$

$H_2 = 9.7 - 1.0 = 8.7$ and $H_2O / H_2 \simeq 0.11$

total mols of gas = 22.05 + 9.7 = 31.75

Gas Composition (mol %) (omitting N_2 in coal)

CO	65.8
CO_2	3.6
H_2	27.4
H_2O	3.1

Gas From Melting Unit: 100 % CO

Amount of C needed in melting unit = 16.6 units (calculated previousl

mols CO formed = $\frac{16.6}{12} = 1.383$

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 6 OF 6

DESCRIPTION	PROJECT NO.	MADE BY DATE	CHECKED BY DATE

Combustion: On the reduction unit the amount of air needed is:

$$\text{Mol CO} = 0.658 \times 31.75 = 20.89$$
$$\text{Mol } H_2 = 0.274 \times 31.75 = \underline{8.70}$$
$$29.59$$

$\therefore$ mols O_2 required = 14.8

mols N_2 = $\frac{79}{21} \times 14.8 = 55.7$

$\therefore$ Total air = 70.5 mol.
(stoich.)

Sulphur in Slag:

Units in Coal = 350 × 0.01 = 3.5
Units in ore = 1502 × 0.0001 = 0.15

Total in Slag = 3.65

$\therefore$ Slag Analysis = $^{3.65}/_{228}$ = 1.6 %

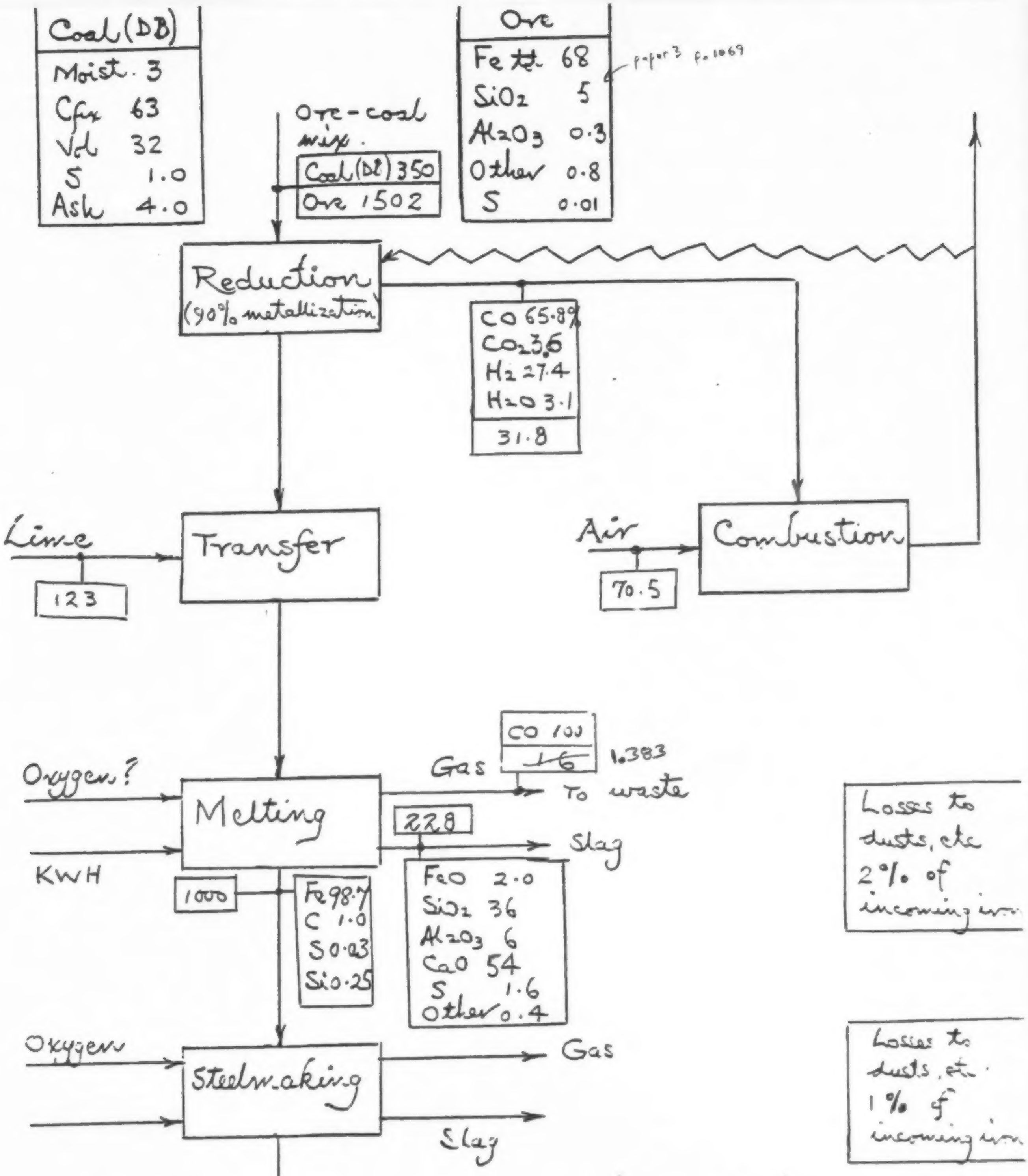
Iron Losses:

melting slag	4.5
melting dusts etc	20.0
steel making dusts etc	10.0
steel making slag @ 30% FeO $= 1000 \times 0.15 \times 0.30 \times \frac{55}{72}$	35.0
	70

Lime to Melting

228kg Slag @ 54% CaO

$\therefore$ Assuming 100% CaO for lime requirement= 123.1 kg / tonne hot metal.


Coal (DB)
Moist. 3
Cfix 63
Vol 32
S 1.0
Ash 4.0
Ore
Fe tot 68
paper 3 p. 1069
SiO2 5
Al2O3 0.3
Other 0.8
S 0.01
Ore-coal mix.
Coal (DB) 350
Ore 1502
Reduction (90% metallization)
CO 65.8%
CO2 3.6
H2 27.4
H2O 3.1
31.8
Lime
123
Transfer
Air
70.5
Combustion
Oxygen?
KWH
Melting
CO 100
~~16~~
1.383
Gas
To waste
228
Slag
1000
Fe 98.7
C 1.0
S 0.03
Si 0.25
FeO 2.0
SiO2 36
Al2O3 6
CaO 54
S 1.6
Other 0.4
Losses to dusts, etc 2% of incoming iron
Oxygen
Steelmaking
Gas
Slag
Losses to dusts, etc. 1% of incoming iron
Steel

Mass units on solids
Mol unit on gases

Approx. overall iron recovery 93%

LB Process - Materials & Energy Diagram.

FIG. 2. Essentials of Materials

AGW_
5006

1.25 GJ

295,688.12

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 1 OF			
DESCRIPTION inputs	PROJECT NO.	MADE BY DATE	CHECKED BY DATE

MW = 231.55

kcal.

Iron Ore @ 25°C (Magnetite - Fe_3O_4)

$\Delta H_{f_{298}} = -267.8$ kcal / gmole.

1502 kg = 1502000 g = 6486.72 gmol — -1,737,143.6

Coal @ 25°C (65% C, 8% H_2O equivalent, 27% C_4H_6)
350 kg

227.5 kg C — –

28 kg H_2O = 1555.56 gmol
$\Delta H_{f_{298}} = -68{,}320$ — -106,275.5

94.5 kg C_4H_6 = 1750 gmol

- D82 CRC — $\Delta H_{f_{298}} \simeq 37$ kcal/gmol — 64,750

Air for Combustion of Reduction Unit Offgas:

70.5 kgmol x 1.1 (10% excess) = 77.55 kgmol
= 77550 gmol

take as N_2 and O_2 — –

HATCH ASSOCIATES	CALCULATION SHEET	SHEET NO. 2 OF

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
Inputs		DATE	DATE

kcal

Lime to melting - look at slag forming components

CaO Mw = 56

$\Delta H_{f_{298}} = -151.8$

we ignore the heat of formation of the slag forming compounds and the slag itself because the reaction heat is so small.

Outputs

Iron - 1000 kg at 1530°C

$\Delta H_{f_{298}} = 0$

ht required to heat from 25°C to 1530°C.

at 2800°F 590 BTU/lb P815 BOHS.

= 327.78 kcal/kg 327,778

Slag - 228 kg @ 1550°C

775 BTu/lb P812 BOHS

= 430.56 kcal/kg 98,167

HATCH ASSOCIATES	CALCULATION SHEET			SHEET NO. 3 OF
DESCRIPTION		PROJECT NO.	MADE BY DATE	CHECKED BY DATE

Offgas from reduction Unit - assume total combustion.

Pre Combustion	Combustion Air	Exit Gas.
Mol CO = 20.89	Mol N_2 = 61.24	Mol N_2 61.24
Mol H_2 = 8.70	Mol O_2 = 16.28	Mol O_2 1.48
Mol CO_2 = 1.145		Mol CO_2 22.035
Mol H_2O = 0.985		Mol H_2O 9.685

		Kcal
N_2	$\Delta H_{F_{298}} = 0$	—
	Ht content at 800°C 5,918 cal/gmol	362,418.3
O_2	$\Delta H_{f_{298}} = 0$	—
	Ht Content at 800°C 6038 cal/gmol	
CO_2	$\Delta H_{F_{298}} = -94.05$ kcal/gmol	-2,072,391
	Ht Content at 800°C 8950 cal/gmol.	197,213.3
H_2O	$\Delta H_{f_{298}} = -57.8$ kcal/gmol	-559,793
	Ht content at 800°C 6970 cal/gmol	67,504.5

Melting Unit Offgas: CO at 1400°C 1.6 kgmol.

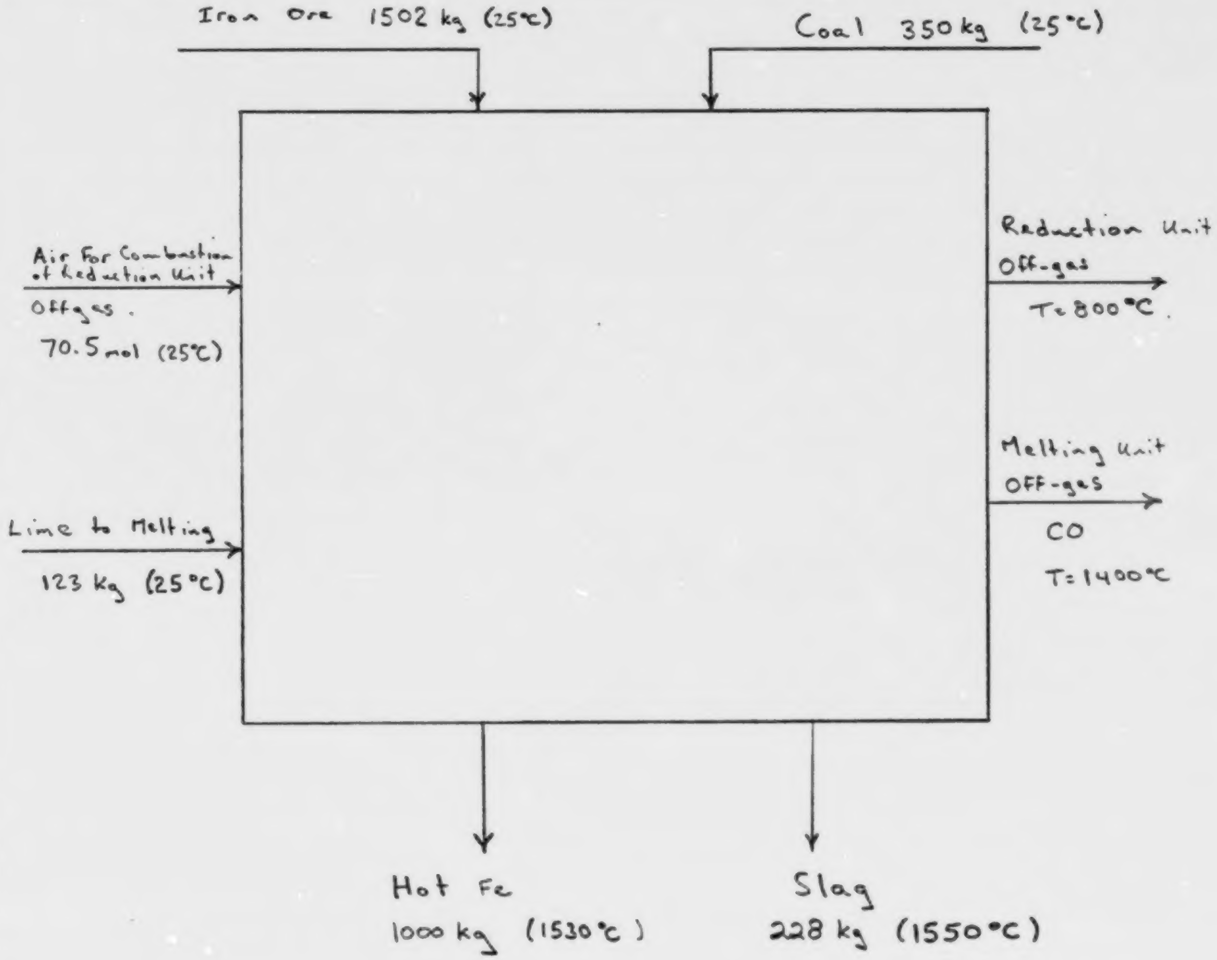
$\Delta H_{F_{298}}$ -26,416 cal/gmol	-42,265.6
Ht content at 1400°C 10850 cal/gmol	17,360.

HATCH
ASSOCIATES

CALCULATION SHEET

SHEET NO. 4 OF

DESCRIPTION	PROJECT NO.	MADE BY DATE	CHECKED BY DATE


Iron Ore 1502 kg (25°C)
Coal 350 kg (25°C)
Air for Combustion of Reduction Unit Offgas.
70.5 mol (25°C)
Lime to Melting
123 kg (25°C)
Reduction Unit Off-gas
T=800°C.
Melting Unit Off-gas
CO
T=1400°C
Hot Fe
1000 kg (1530°C)
Slag
228 kg (1550°C)

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. ...1... OF ....

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
Reduction Unit Energy Balance		DATE	Nov. 2/87 DATE

Kcal

INPUTS : Iron Ore 1502 kg @ 25°C

kg-mol

6.092	1410.4 Kg Fe_3O_4	$\Delta H_{F298} = -267.8$ kcal/g-mol	-1,631,438
1.25	75.1 Kg SiO_2	$\Delta H_{F298} = -217.6$ kcal/g-mol	
0.1618	16.5 Kg Other (Al_2O_3)	$\Delta H_{F298} = -399.6$ kcal/g-mol	

Coal @ 25°C (65% fixed C , 8% H_2O, 27% C_4H_6) 350 kg

C - 227.5 kg -

H_2O 28 Kg = 1555.56 g-mol

$\Delta H_{f298} = -68,320$ cal/g-mol -106,275.6

C_4H_6 94.5 kg = 1750 g-mol

$\Delta H_f =$ 37 kcal/g-mol 64,750

Combustion Air 77.55 kg-mol

N_2	61.24 kg-mol	$\Delta H_{f298} = 0$
O_2	16.28	$\Delta H_{f298} = 0$

Total Inputs -1,672,963.6

HATCH ASSOCIATES	CALCULATION SHEET	SHEET NO. 2 OF

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
Reduction Unit Energy Balance		DATE	DATE Nov 2/87

Outputs (90% metallization) 1100 °C

Fe	919.3 kg		$\Delta H_{f298} = 0$	
FeO	102.1 kg		$\Delta H_{f298} = -63,800$ cal/g-mol	-90,660.8
SiO_2	75.1 Kg	1250 gmol	$\Delta H_{f298} = -217,600$ cal/g-mol	
Al_2O_3 (other)	16.5 Kg	161.8 g-mol	$\Delta H_{f298} = -399,600$ cal/g-mol	

71.85

Heat Content @ 1100 °C

Fe	10,120 cal/g-mol	× 16.46 kg-mol	166,576.8
FeO	14520 cal/g-mol	× 1.421 kg-mol	20,633.2
SiO_2	22270 cal/g-mol	× 1.25 kg-mol	27,837.5
Al_2O_3	30,850 cal/g-mol	× 0.1618 kg-mol	4,991.5

Offgas

	kg-mol	Ht Content cal/g-mol	ΔH_{f298} kcal/g-mol	
N_2	61.24	5918	0	-362,415.3
O_2	1.48	6038	0	-8,936.2
CO_2	22.035	8950	-94.05	-2,072,391.0 197,213.3
H_2O	9.685	6970	-57.8	-559,793.0 67,504.5

TOTAL Outputs -1,866,733.5

TOTAL Inputs -1,672,963.6

Surplus Energy 193,769.9

if heat losses = 10% Surplus Energy 7,096.6

if heat losses = 20% Surplus Energy -179,576.8 (energy required [illegible]

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 3 OF

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
Melting Unit Energy Balance		DATE	DATE Nov 2/87

Transfer : Output of reduction unit cools from 1100°C to 900°C

Melting Unit Inputs

			kcal
Fe	919.3 kg	$\Delta H_{f_{298}} = 0$	
FeO	102.1 kg	ΔH_{f298} = −63,800 cal/gmol	− 90,660.8
SiO_2	75.1 kg	ΔH_{f298} = −217,600 cal/gmol	
Al_2O_3 (+ other)	16.5 kg	ΔH_{f298} = −399,600 cal/gmol	

Heat Content at 900°C

Fe	8,043.3 cal/gmol x 16.46 kg mol	132,392.7
FeO	11,700 cal/gmol x 1.421 kg mol	16,625.7
SiO_2	13,610 cal/gmol x 1.25 kg mol	17,012.5
Al_2O_3	20,170 cal/gmol x 0.1618 kg mol	3,263.5

Lime at 25°C - The heat of formation of the slag forming compounds is ignored because it is small compared to the other reactions

Total inputs 78,633.6

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. 4 OF

DESCRIPTION Melting Unit Energy Balance.

PROJECT NO.

MADE BY

DATE

CHECKED BY

DATE Nov.2/87

		Kcal
Outputs.	Iron at 1530 °C	
	Heat Content 327.78 Kcal/Kg × 1000 Kg	327,780
	Slag at 1550 °C	
	Heat Content 430.56 Kcal/Kg × 228 Kg	98,167
Offgas	CO at 1400 °C $\Delta H_{298} = -26,416$ cal/gmol × 1.6 kgmol	-42,265.6
	Heat content at 1400 °C 10,850 cal/gmol	17,360
	Total Outputs	401,041.4 kcal
	Inputs	78,633.6
	Total Requirements	322,407.8 kcal
	allowing 20% losses Requirement =	386,889.4 kcal
		(1,535,393.2 BTu)

HATCH ASSOCIATES	CALCULATION SHEET			SHEET NO. 1 OF 2
DESCRIPTION Reduction Unit		PROJECT NO.	MADE BY DATE	CHECKED BY DATE Nov. 3/87

Inputs	Fe_3O_4	1410.4 kg			-1,631,438
	SiO_2	75.1 kg			
	Other	16.5 kg			
Coal	C -	227.5 kg			
	H_2O -	28 kg			-106,275.6
	C_4H_6	94.5 kg			64,750
				Total Inputs	-1,672,963.6
Outputs	Fe	- 919.3 kg			
	FeO	- 102.1 kg			-90,660.8
	SiO_2	- 75.1 kg			
	Al_2O_3	- 16.5 kg			

Heat Content @1100°C	Fe				166,576.8
	FeO				20,633.2
	SiO_2				27,837.5
	Al_2O_3				4,991.5
			kg mol	ΔH_{f298}	
Offgas heat content	CO	6572	20.89	-26.42	137,289.1 -551,913.8
	CO_2	8950	1.145	-94.05	10,247.75 -107,687.25
	H_2	6221.9	8.70	0	54,130.53
	H_2O	6970	0.985	-52.8	6,865.45 -56,933.0

Total Outputs -270,935.8

heat requirement 1,402,027.8 kcal

HATCH ASSOCIATES	CALCULATION SHEET			SHEET NO. 2 OF 2
DESCRIPTION Combustion Unit		PROJECT NO.	MADE BY DATE	CHECKED BY DATE Nov. 3 /87

		kg-mol	ΔHf		
Inputs: Offgas	CO	20.89	-26.42		- 551,913.8
	CO_2	1.145	-94.05		- 107,687.25
	H_2	8.70	0		-
	H_2O	0.985	-57.8		- 56,933.0
		cal/g-mol			
Sensible heat	CO	6572			137,289.1
	CO_2	8950			10,247.75
	H_2	6221.9			54,130.53
	H_2O	6970			6,865.45
Combustion Air	N_2	61.24 kg-mol			-
	O_2	16.28 kg-mol			-
				Total	-508,001.22
		kg-mol	ΔHf		
Outputs	N_2	61.24	-		-
	O_2	1.48	-		-
	CO_2	22.035	-94.05		- 2,072,391
	H_2O	9.685	-57.8		- 559,793
Sensible Heat @ 800°C	N_2	5918 cal/g-mol			362,418.3
	O_2	6038 cal/g-mol			197,213.3
	CO_2	8950 cal/g-mol			67,504.5
	H_2O	6970 cal/g-mol			17,360
				Total Output	- 1,987,688.2
				Differance	- 1,479,687
Sensible Heat @ 1500°C	N_2	11,385.1 cal/g-mol.			697,223.5
	O_2	11,980.05 cal/g-mol.			17730.47
	CO_2	18,298.5 cal/g-mol.			403,207.45
	H_2O	14,590 cal/g-mol			141,304.15
				Total	1,259,465.57

Heat recovered from Offgas = 1,259,465.57 - 644,496.1 = 614,969.47 kcal

APPENDIX B LB Process Reaction Rates From Muffle Furnace Experiments

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO.
........OF........

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
Muffle Furnace Experiments Oct. 26/87		DATE	DATE

Exp. #5.

20/80 coal/ore mix

Time	Centreline Temp. °C	ΔT
0	25	65
5	90	50
10	140	220
15	360	135
20	495	145
25	640	150
30	790	100
35	890	50
40	940	55
45	995	155
50	1150	90
55	1240	50
60	1290	

crucible 8.3cm I.D.
11cm high.
preheat time = 5 minutes.
heating time = 55 minutes.

From the above data we see that once the material reaches the temperature range 790-890°C (at ~ 32.5 minutes), the centre line temperature does not increase very rapidly until the 45 minute mark. The rate of change of the centreline temperature remains constant at ~50°C/5. This is due to the Boudouard reaction taking place which consumes a major portion of the heat supplied. This occurs until the material has reacted at which point the sponge continues to heat up. An interesting point to note is that the sponge does not heat up any faster than the ore/c mixture.

distance reaction front must travel = 8.3cm/2 = 4.15 cm
time for reaction front to hit centreline
and react material = 45 minutes. - 5 minutes preho
= 40 minutes
∴ Effective reaction rate = 4.15/40 = 0.404 cm/min.

HATCH
ASSOCIATES

CALCULATION SHEET

DESCRIPTION	PROJECT NO.	MADE BY DATE	SHEET NO. .... OF .... CHECKED BY DATE

time required to penetrate a 30 cm bed = $30/.104$ = 289.16 minute

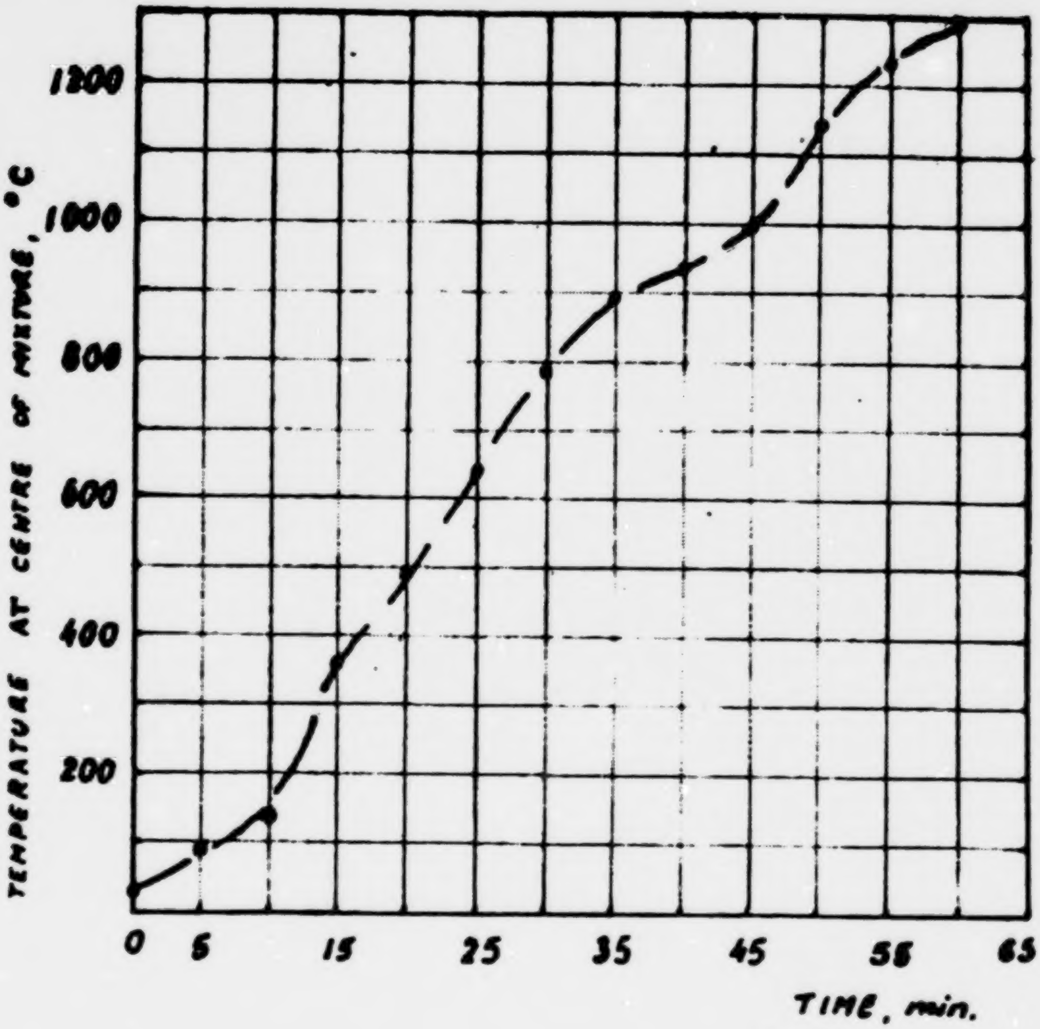
Scale up factor to keep reactor volume/surface area ratio constant

$$= 3.61 \quad \left(\frac{30\text{cm ID}}{8.3\text{ cm ID}} \right)$$

$\therefore$ residence time = 3.61×289.16 = 1045.16 min
= 17.42 hours

For vertical unit - scale up factor = 3.61
effective bed thickness = $30\text{cm}/2$ = 15cm

$\therefore$ required residence time $\simeq$ 8.7 hours.



TEMPERATURE OF CENTRE OF LOADED CRUCIBLE AS FUNCTION OF TIME FOR SiC CRUCIBLE, EXP. # 5.

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. .... OF			
DESCRIPTION Muffle Furnace Experiments	PROJECT NO.	MADE BY DATE	CHECKED BY DATE

Nov. 10, 1987. Experiment #1

The tip of the thermocouple was located 10cm from the hot face. After approximately 80 minutes the rate of temperature increase at the thermocouple decreases. The reason given is that heat losses are occuring at the end of the SiC tube. However the tube wall would be hotter than the material in the bed and thus heat would not flow from the ore/coal mix to the tube walls and out of the system.

It is interesting to note that at approximately the same time as the tempe tu starts to level out, heavy smoke started to flow from the cold end. The experiment state that the depth of material completely metallised over a period of 115 minut s was 8cm or an effective rate of .7mm/minute. Our examination of the material indicat that there was still a considerable amount of coal present in the material that was supposed to be 100% metallised.

Experiment #2

In this experiment the temperature measured by the thermocouple once agai started to level out after 60-70 minutes. The temperature at which phenom na started to occur was once again in the 450-500°C range. Thus it is likely that factors other than heat losses were responcible. The ore/coal mixture was metallised to a depth of 4.5cm over a period of 90 minutes. This corresponds to a rate of $4.5\text{cm}/90\text{ minutes} = 0.05\text{cm}/\text{min} = .5\text{mm}/\text{min}$.

It should be noted that in both of these experiments heat transfer occurred by radiation to the ore/coal mixture exposed at the uninsulated e of the tube. It is likely that as the CO left the tube at this end it b n and produced local temperatures much in excess of the furnace tempera which was in the range 1000-1400°C. This is not likely to be representativ . the heat transfer through the tube to the material though it is a good representation of the heat transfer occuring at the end of the reactor t in the bench-scale process.

HATCH ASSOCIATES	CALCULATION SHEET			SHEET NO. .... OF ....
DESCRIPTION		PROJECT NO.	MADE BY DATE	CHECKED BY DATE

Experiment #3

This experiment was carried out in a tube placed on its end with a cover pl on the top of the tube. Thus there was no direct radiative heat transfer to the coal/ore mixture. The investigators found that the material was metallized to a depth of 2.0 cm over a period of 25 minutes. This gives an effective reaction rate of 0.8 cm/min which is even faster than for the case when the materia is directly exposed (experiments 1 and 2). This is rather unlikely since the exposed material would received heat by radiation and also would receive the added benefit of the CO combustion.

Experiment #4.

This experiment was carried out in the same manner as experiment #3 except that the tube was flipped on its side after 25 minutes in order to expose the ends for the last 10 minutes of heating. The ~~top~~ material at the top of the tube metallized to a depth of ~1.0 cm while that on the bottom of the tube (which had received ~~not~~ heat for the first 25 minutes) was metallized to a depth of ~0.5 cm. It can be concluded then that the rate of reaction was ~0.5 mm/min for the last 10 minutes when heat transfer was due to radiation. Once again CO combustion would give considerable benefit. Based on this rate of reaction over the last 10 minutes, the rate of reaction for the first 25 minutes may be obtained by difference (ie depth reacted at top face - depth reacted at bottom face)/1 Thus an effective reaction rate of 0.2 mm/min is obtained for the first 25 minut of heating.

November 12, 1987
W-K. Lu

EXPERIMENTS IN A MUFFLE FURNACE

(a) Seven experiments were carried out in October 1987. Experimental results have been presented and reacted specimens shown at the meeting on October 26, 1987 in Toronto. Iron ore used in these experiments was magnetite concentrate from Northern Ontario.

(b) Due to the fact that almost all Canadian iron ores are produced on the north shore of the St. Lawrence and are hematite, fines from Wabush mines were used in experiments carried out on November 10 and 11, 1987. Properties of ore and coal are documented on the following page.

(c) November 10, 1987 Experiments: Growth of metal shell under very unfavourable conditions.

A one-dimensional reacting system was placed at the spot where the door used to be. Crucibles were wrapped heavily in insulation materials, except one end which faced the chamber of the furnace. Details are shown on the enclosed figures. To the best of our estimation, the depth of mixtures which are completely metallized is at least 8 cm in Experiment #1 and 4.5 cm in Experiment #2.

(d) November 11, 1987 Experiments: Growth of metal shell and the effect of direct radiation.

Details of Experiments #3 and #4 are shown on the enclosed figures. Taking note of the fact that it takes ten minutes for heating up the crucible to initiate the reactions, the depth of the metal shell is about 20 mm in 25 minutes in Experiment #3. Under otherwise the same experimental conditions, at minute 25 in Experiment #4 the cover was removed and the crucible turned 90°, direct radiation through both ends results in a much higher degree of metallization of the specimen. Even though, the range of temperature in Experiments #3 and #4 is rather on the low side.

RAW MATERIALS USED IN EXPERIMENTS CONDUCTED

NOVEMBER 10 and 11, 1987

(1) Wabush Fine, Received from Wabush Mines in November 1985

Chemical Analysis 65.26% Fe
2.5% SiO_2
1.83% Mn

Size Analysis

+ 20 mesh (850 m)	0 wt.%
- 20 mesh + 30 mesh (600 m)	9.3 "
- 30 mesh + 50 mesh (300 m)	26.1 "
- 50 mesh + 100 mesh (150 m)	29.4 "
- 100 mesh	35.2 "

It was used as received.

(2) High Volatile Coal, Received from Dofasco in 1987

Volatile Matter 30.6%

Ash in the range of 6 to 7%

Size Analysis

	As-Received	Used in Experiment
+ 20 mesh	45.8 wt.%	0 wt.%
- 20 mesh + 30 mesh	8.1 "	11.9 "
- 30 mesh + 50 mesh	15.7 "	23.2 "
- 50 mesh + 100 mesh	12.5 "	18.4 "
- 100 mesh	17.9 "	46.4 "

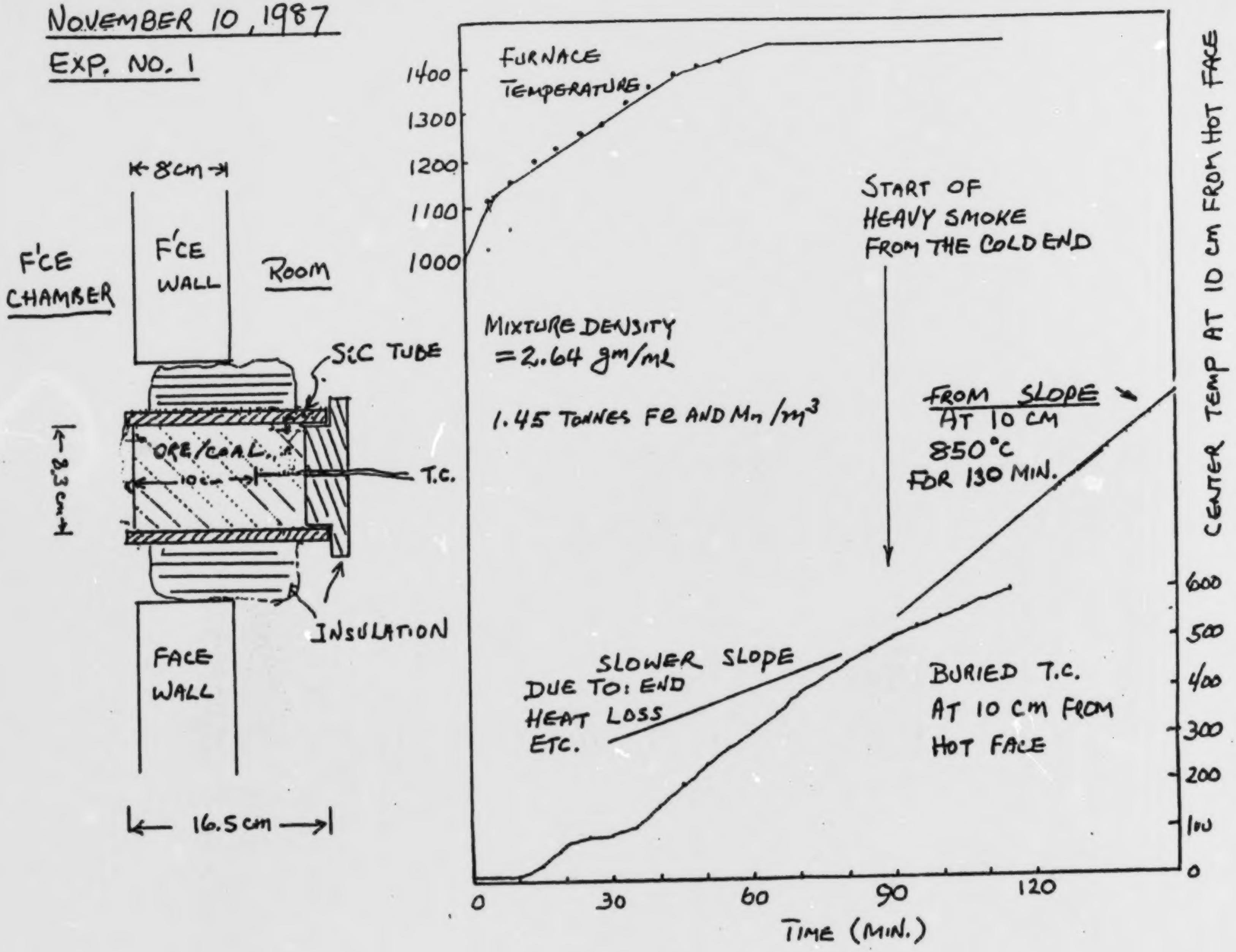
About 50% of coal needs grinding.

(3) Coal/ore mixture has 18% coal and 82% ore, by weight.

NOTE: Three high volatile coals have been used. They were supplied by Dofasco Inc., Stelco Inc. and Ontario Hydro (thermal coal). No difference was detected in our experiments.

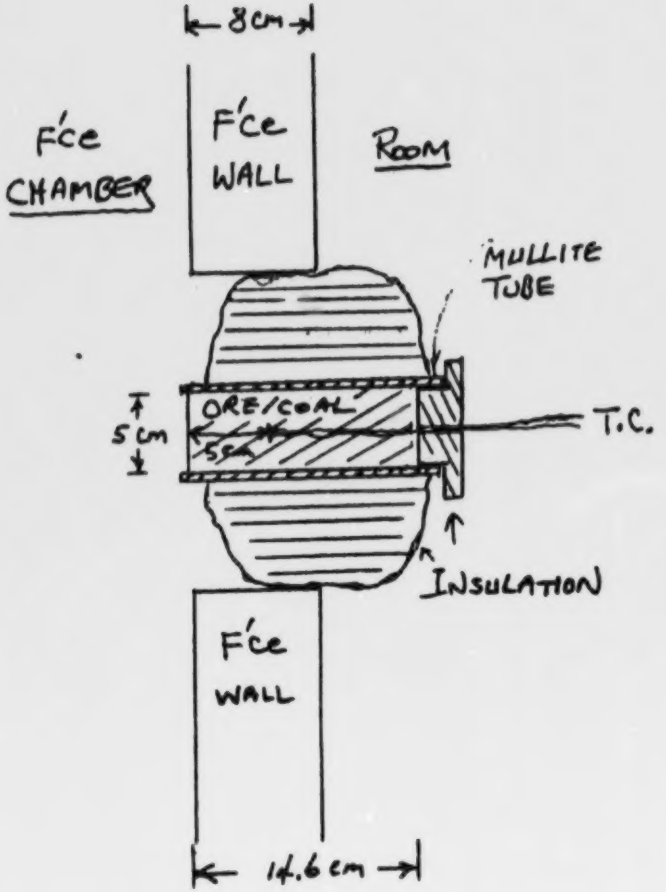
NOVEMBER 10, 1987

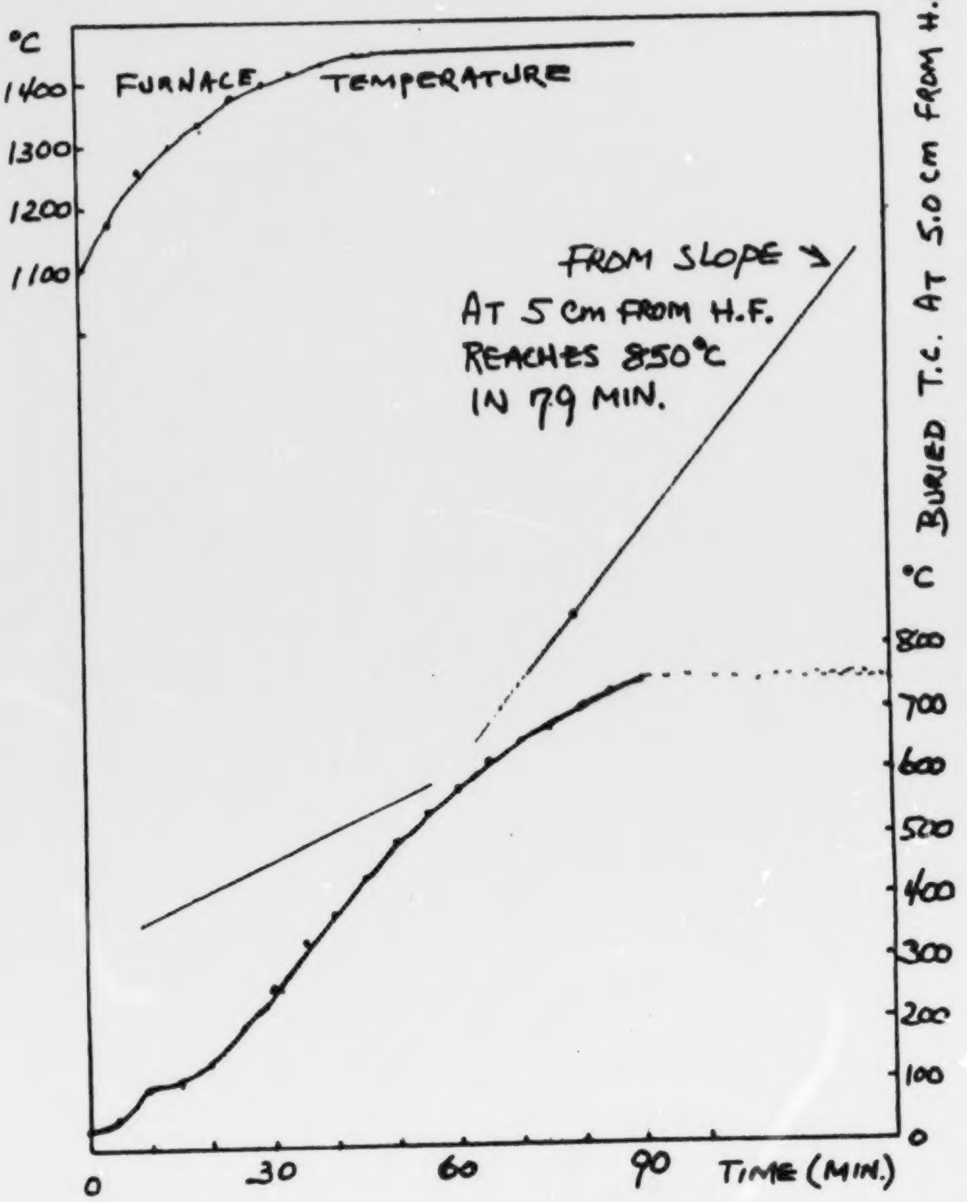
EXP. NO. 1



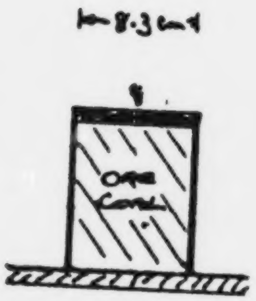
NOVEMBER 10, 1987

EXP. NO. 2


F'ce CHAMBER
← 8cm →
F'ce WALL
ROOM
MULLITE TUBE
5 cm
ORE/COAL
5cm
T.C.
INSULATION
F'ce WALL
← 14.6 cm →



NOVEMBER 11, 1987

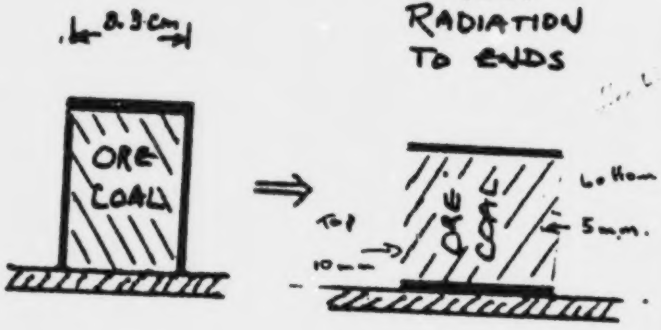

8.3 cm
ORE
COAL

~ 20 cm over 25 minutes.

Exp. # 3. MIN. 0 - 35

10 min before reaction starts.

Exp # 4


8.3 cm
ORE
COAL
⇒
Top
10 mm
ORE
COAL
bottom
5 mm.

DIRECT RADIATION TO ENDS

MIN. 0 -25

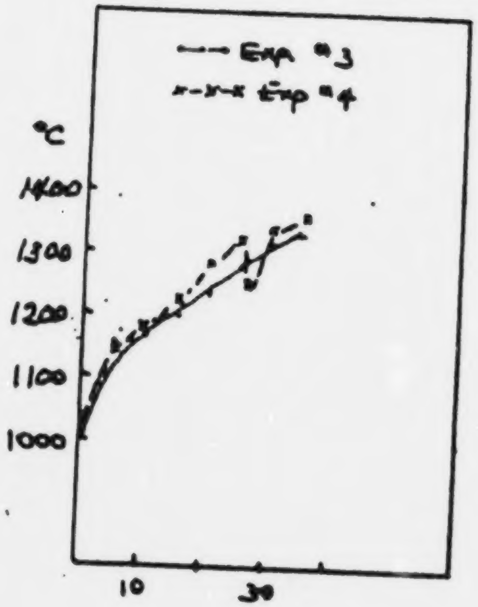
MIN. 26 -35

radiation at ~ 1300 °C

~ 10 minutes.

BLACK SMOKE STARTS (CARBONIZATION)

MIXTURES REACH 500-600°C



AT MIN. 20
F'ce DOOR OPENED
TOP COVER REMOVED
CRUCIBLE TURNED 90°

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CALCULATION SHEET

DESCRIPTION Notes: LB Experiments Nov. 17/87.	PROJECT NO.	MADE BY DATE	SHEET NO. ....OF........ CHECKED BY DATE

preheating of the mixture to 700°C only cut the residence time by 5min
∴ concluded that heat transfer in the ~~coarse~~ unreacted mixture is no
important in the overall Kinetics.

- suggests that for a 15cm thick bed a residence time of 70 min will suffice.

- base all of conclusions on penetration of reaction front → assume 100
reduction once the reaction front has penetrated past a point → may onl
be 80-90% (They say themselves that in 100% metallized product
they can tap it and coal dust comes out)

Experiment #1 November 17, 1987

It is interesting to note here how rapidly the ore/coal mixture heats up. The material at thermocouple #2 heats up from 0°C to 300°C in just 10 minutes and to 605 degrees in 15 minutes. The heating was stopped after 55 minutes at which time the reaction front had proceeded 5cm into the tube. This gives an effective reaction rate of ~0.9mm/minute. Heat transfer was by radiation to the open end of the tube and by radiatio
to the mullite tube walls and then axial conduction to the ore/coal bed.

Experiment #2

The coal ore mix was preheated at 700°C for 1 hour prior to starting the experiment. In this case it took only 45 minutes for the reaction front to penetrate 5cm; a time saving of 10 minutes or $\frac{10}{55}$ = 18.2%

HATCH ASSOCIATES

CALCULATION SHEET

DESCRIPTION	PROJECT NO.	MADE BY DATE	SHEET NO. .... OF .... CHECKED BY DATE

Experiment 3

Heat transfer was by radiation to the open end of the tube. The reaction front travelled 5cm in 90 min or 0.55 mm/min. Thus we can probably safely assume that in experiment #1 where the effective rate was 0.9mm/min, the axial conductive heat transfer was responsible for a rate of 0.34mm/min (ie 61% of that due to radiation).

Mass and Energy Balances compare quite well with what we obtained.

at 99.7% reduction the energy requirements are 3.17Gcal/THM
13.27 GJ/THM

at 95% reduction the energy requirements are 3.25 Gcal/THM
13.60 GJ/THM

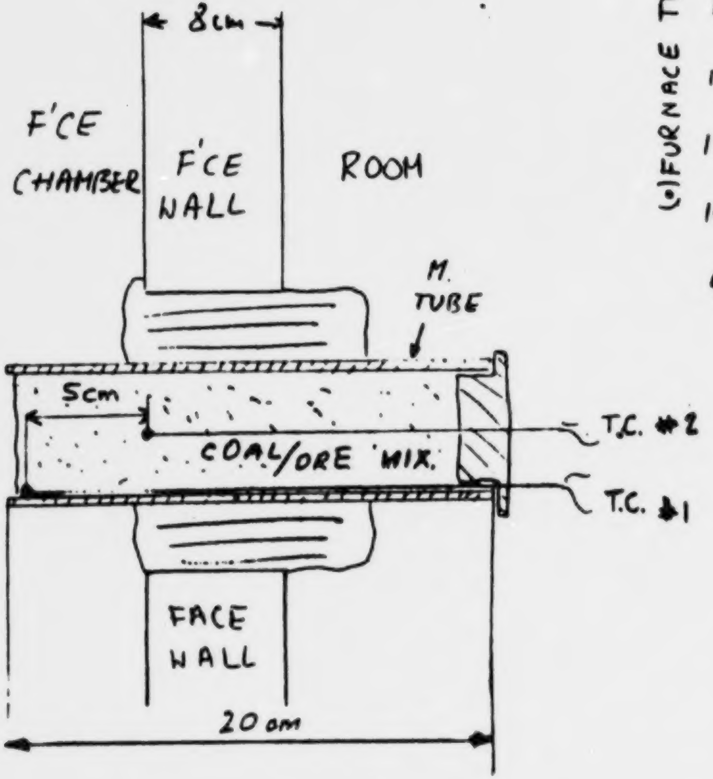
at 80% reduction the energy requirements are 3.54 Gcal/THM.
14.82 GJ/THM.

essentially for every additional 5% of reduction that is not performed in the reactor and must be performed in melting unit the electrical energy requirement increases 0.085 Gcal/THM.

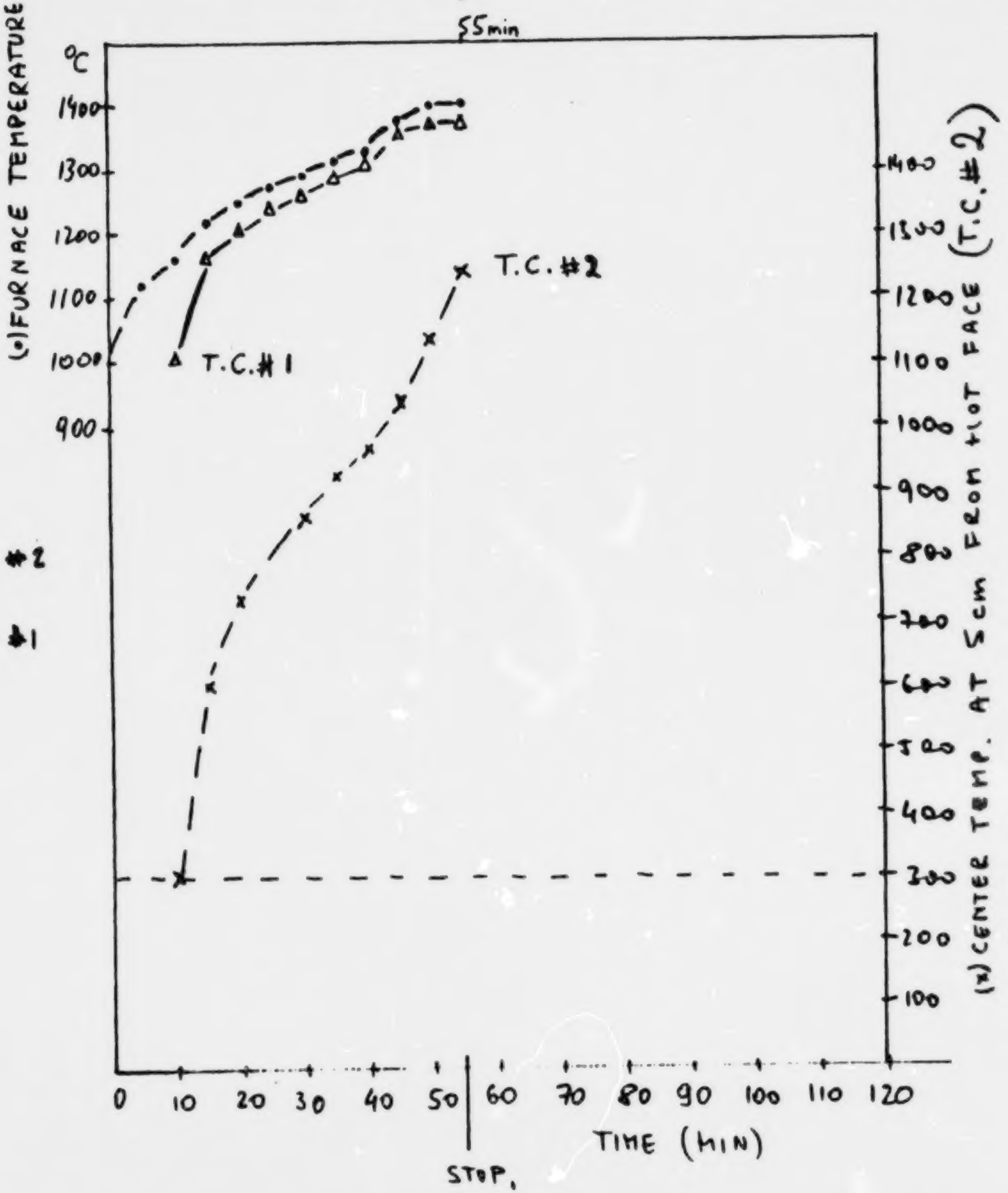
[illegible] tal [illegible] to [illegible] .9 [illegible] min

NOVEMBER 17, 1987

Exp. No.1.



8cm
F'CE CHAMBER
F'CE WALL
ROOM
M. TUBE
5cm
COAL/ORE MIX.
T.C. #2
T.C. #1
FACE WALL
20 cm



APPENDIX C Heat Flow Through Porous Material

FMW/[redacted]/RS

HATCH ASSOCIATES LTD. CONSULTING ENGINEERS

HATCH
ASSOCIATES

HP9000 PGM: [redacted]

HEAT FLOW THROUGH POROUS MATERIAL

: BASIC EQNS

HP9000 PRGM: POR_MTL

JOB # : 5008

/87/11/12/
/13/

THERMAL MODEL - PORES

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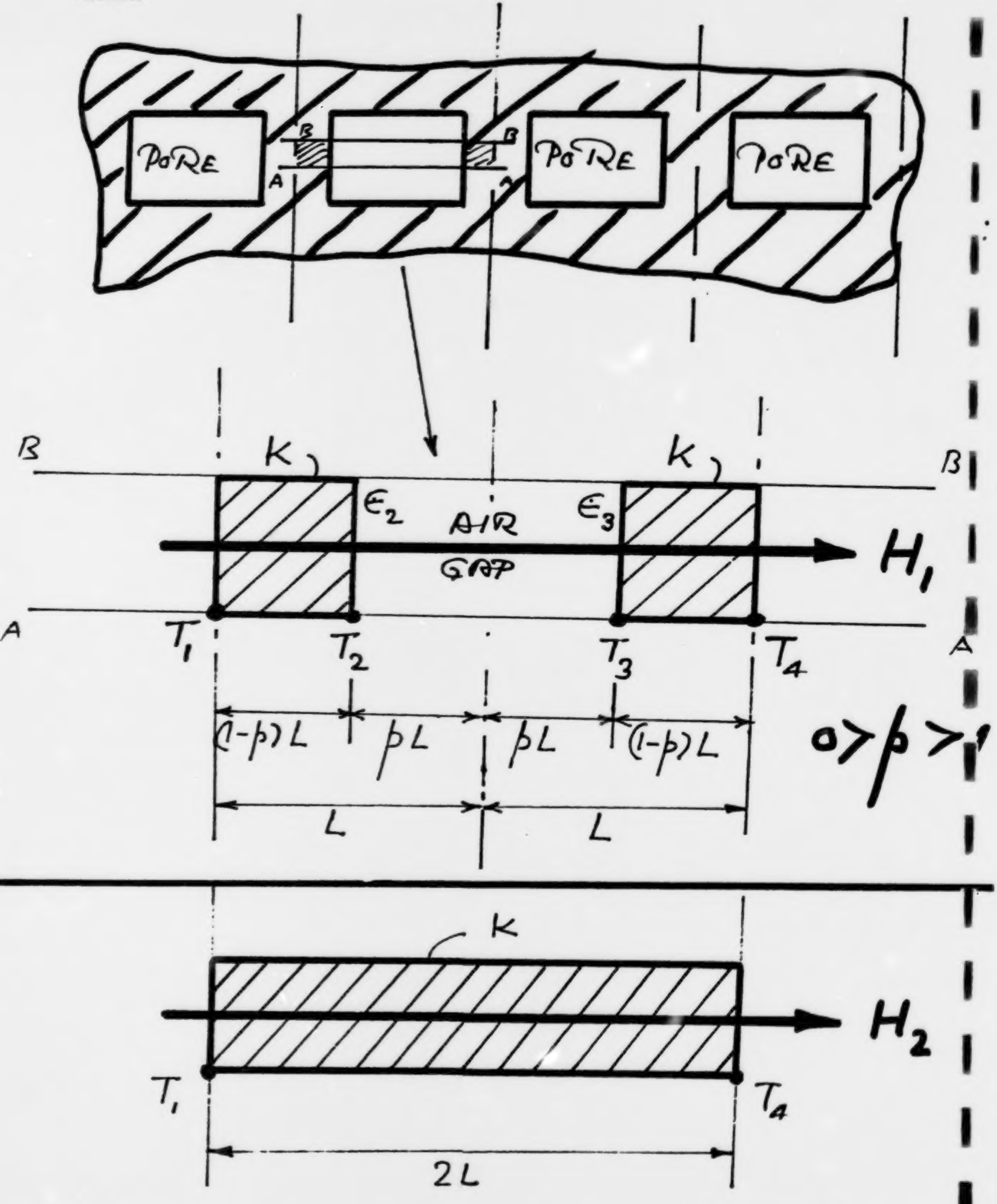

PORE
B
A
B
A
PORE
PORE
B
K
ε2
AIR GAP
ε3
K
B
H1
A
T1
T2
T3
T4
A
(1-p)L
pL
pL
(1-p)L
0>p>1
L
L
K
H2
T1
T4
2L

MODEL TO COMPARE: W/O PORE

RNK&P
F3/11

FLUX THRU' MTL WITH PORE :

$$H_1 = KA.\frac{T_1 - T_2}{(1-p)L} = \left[\frac{KA}{(1-p)L}\right]^{\cdot}\cdot(T_1 - T_2)$$

$$T_1 - T_2 = H_1 * \frac{1}{[\ ]^{\cdot}} \qquad \text{———} \qquad (1)$$

$$H_1 = 0.173\times A\times\underbrace{\left[\frac{1}{\frac{1}{\epsilon_2}+\frac{1}{\epsilon_3}-1}\right]}_{\bar{\epsilon}\ (\text{say})}\times\left[\left(\frac{T_2+460}{100}\right)^4 - \left(\frac{T_3+460}{100}\right)^4\right]$$

$$= \left[0.173 * A * \bar{\epsilon} * \frac{\left[\left(\frac{T_2+460}{100}\right)^4 - \left(\frac{T_3+460}{100}\right)^4\right]}{(T_2 - T_3)}\right]^{\cdot\cdot}\cdot(T_2 - T_3)$$

$$T_2 - T_3 = H_1 * \frac{1}{[\ ]^{\cdot\cdot}} \qquad \text{———} \qquad (2)$$

$$H_1 = KA.\frac{T_3 - T_4}{(1-p)L} = \left[\frac{KA}{(1-p)L}\right]^{\cdot\cdot\cdot}\cdot(T_3 - T_4)$$

$$T_3 - T_4 = H_1 * \frac{1}{[\ ]^{\cdot\cdot\cdot}} \qquad \text{———} \qquad (3)$$

ADDING (1), (2) & (3) :

$$T_1 - T_4 = H_1\cdot\left\{\frac{1}{[\ ]^{\cdot}} + \frac{1}{[\ ]^{\cdot\cdot}} + \frac{1}{[\ ]^{\cdot\cdot\cdot}}\right\}$$

87/11/12

$$H_1 = \frac{T_1 - T_4}{\frac{1}{[\ ]^{\cdot}} + \frac{1}{[\ ]^{\cdot\cdot}} + \frac{1}{[\ ]^{\cdot\cdot\cdot}}} \qquad \text{——} \quad (4)$$

WHERE:

$$[\ ]^{\cdot} = \frac{KA}{(1-p)L}$$

$$[\ ]^{\cdot\cdot\cdot} = \frac{KA}{(1-p)L}$$

$$[\ ]^{\cdot\cdot} = 0.173 * A * \bar{\epsilon} * \frac{\left(\frac{T_2 + 460}{100}\right)^4 - \left(\frac{T_3 + 460}{100}\right)^4}{(T_2 - T_3)}$$

$$\bar{\epsilon} \longrightarrow = \frac{1}{\left(\frac{1}{\epsilon_2} + \frac{1}{\epsilon_3} - 1\right)}$$

$$H_2 = \frac{T_1 - T_4}{2L} \cdot KA \qquad \text{——} \quad (5)$$

8 7/11/72

$$\frac{\text{HEAT FLOW THRU 'POROUS' MATL}}{\text{HEAT FLOW THRU 'SOLID' MATL}} = \frac{H_1}{H_2} = \frac{(4)}{(5)}$$

$$= \frac{\cancel{(T_1 - T_4)}}{\left\{ \frac{1}{[\ ]^{\cdot}} + \frac{1}{[\ ]^{\cdot\cdot}} + \frac{1}{[\ ]^{\cdot\cdot\cdot}} \right\} \cdot \cancel{(T_1 - T_4)} \cdot \frac{KA}{2L}}$$

$$= \frac{1}{\left\{ \frac{(1-p).L}{KA} + \frac{1}{[\ ]^{\cdot\cdot}} + \frac{(1-p).L}{KA} \right\} \cdot \frac{KA}{2L}}$$

$$= \frac{1}{\left\{ \frac{1}{2}(1-p) + \frac{1}{2}(1-p) + \frac{1}{\frac{2L}{K\cancel{A}}(0.173)A.\bar{\epsilon}\,\frac{\left(\frac{T_2+460}{100}\right)^4 - \left(\frac{T_3+460}{100}\right)^4}{(T_2 - T_3)}} \right\}}$$

$$= \frac{1}{(1-p) + \frac{1}{\frac{0.346.L.\bar{\epsilon}}{K} \cdot \frac{\left(\frac{T_2+460}{100}\right)^4 - \left(\frac{T_3+460}{100}\right)^4}{(T_2 - T_3)}}}$$

87/11/12

$$\rightarrow = \frac{1}{(1-p) + \frac{K}{0.346\,L\,\bar{\epsilon}} \cdot \frac{T_2 - T_3}{\left[\left(\frac{T_2+460}{100}\right)^4 - \left(\frac{T_3+460}{100}\right)^4\right]}}$$

$$\bar{\epsilon} = \frac{1}{\left(\frac{1}{\epsilon_2} + \frac{1}{\epsilon_3} - 1\right)}$$

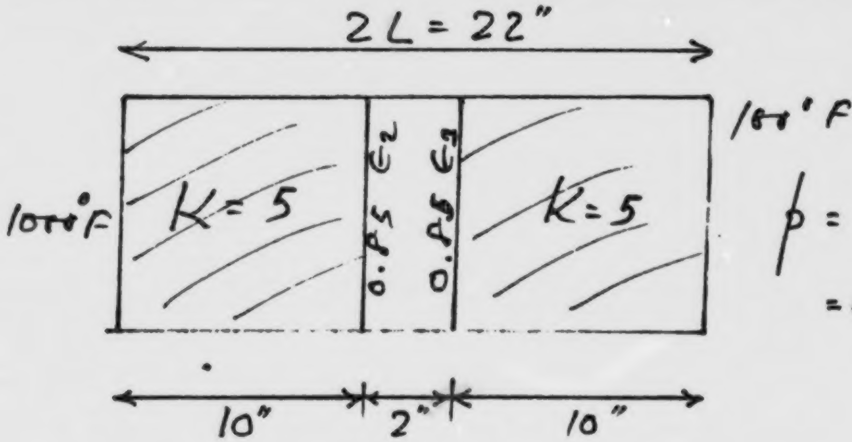
THIS IS RATIO OF HT. FLOW TROUGH POROUS MTL TO HT. FLOW THROUGH NON-POROUS MTL !

NOTE: T_2 AND T_3 ARE 'FUNCTIONS' OF → p, K, L, ϵ_2 & ϵ_3 !
HENCE THE COMPLEXITY !

87/11/13

VERIFY 'POR_MTL' RESULTS

(Example 1)



$p = \frac{2}{22}$

$= 0.09090909$

FOR SOLID MTL] $H_2 = 5 \times 1 \times \frac{1000-100}{22/12} = 2454.54$

(POR_MTL : 2454.55 ✓)

FOR POROUS MTL] POR_MTL : [illegible]

$H_1 : 1736.07$
$T_2 : 710.66$
$T_3 : 389.34$] Are these O.K. ?

Hand Calc :

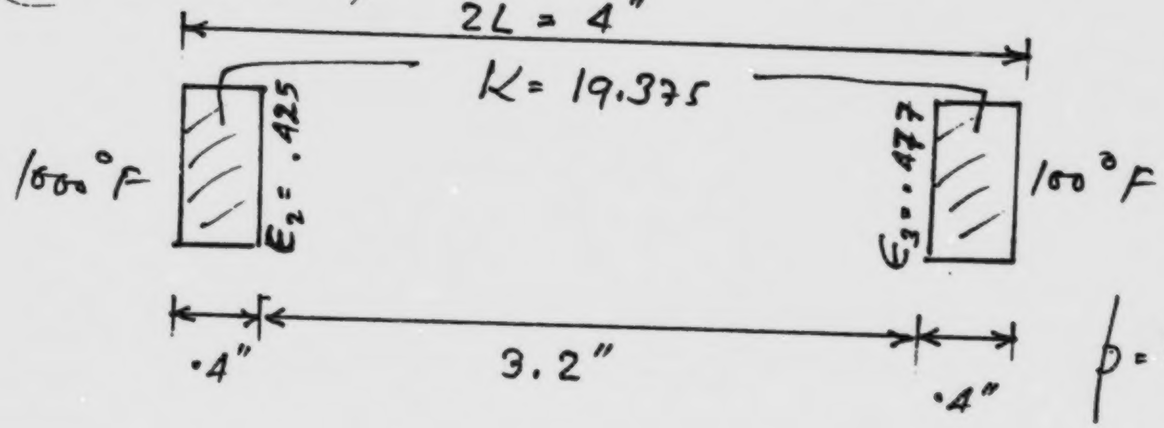
$$\dot{H}_1 = 5 \times \frac{1000 - 710.66}{10/12} = 1736.04 ✓$$

$$\dot{H}_3 = 5 \times \frac{389.34 - 100}{10/12} = 1736.04 ✓$$

$$\dot{H}_2 = .173 \times \frac{1}{\frac{1}{.85} + \frac{1}{.85} - 1} \times \left[\left(\frac{710.66+460}{100}\right)^4 - \left(\frac{389.34+460}{100}\right)^4 \right]$$

$= .173 \times .73913 \times (18781.19 - 5203.569) = 1736.12$ ✓

(Example 2)



$p = \frac{3.2}{4}$

$= 0.8$

$\boxed{H_2 \text{ (Solid Mtl)}}$

Hand Cal : $Flx > 19.375 \times \frac{1000-100}{4/12} = 52312.5$

POR_MTL : $Flx > 52312.5$ ✓

$\boxed{H_1 \text{ (Porous Mtl)}}$

POR_MTC : $T_2 = 996.2071508$

$T_3 = 103.792849$

$Flx = 2204.59$

Hand Cal :

$\overset{\circ}{H_1} = 19.375 \times \frac{1000 - 996.2071508}{.4/12} = 2204.59$ ✓

$\overset{\circ}{H_3} = 19.375 \times \frac{103.792849 - 100}{.4/12} = 2204.59$ ✓

$$\overset{\circ}{H_2} = .173 \times \frac{1}{\frac{1}{.425} + \frac{1}{.477} - 1} \times \left[\left(\frac{996.2071508 + 460}{100}\right)^4 - \left(\frac{103.792849 + 460}{100}\right)\right]$$

$= .173 \times 0.2899074 \times (44966.8678 - 1010.3649) = 2204.59$ ✓

HATCH ASSOCIATES

CALCULATION SHEET

DESCRIPTION	PROJECT NO.	MADE BY DATE	CHECKED BY DATE	SHEET NO. ....OF........

hot face temp. = 2732°F cold face temp = 2710°F.

$\mathcal{E}_2 = \mathcal{E}_3 = 0.65$ semi pattern length = 0.1 inches.

Ratio of Heat Flow Through Porous Material to Heat Flow Through Solid Mater

	$P = 1\times10^{-6}$	0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8	0.9	1.0
K= 25	0.03451	0.03463	0.03475	0.03488	0.035	0.03512	0.03524	0.03537	0.03549	0.03562	0.03575
10	0.08204	0.08272	0.08341	0.08411	0.08482	0.08565	0.08629	0.08704	0.0878	0.08858	0.08937
1	0.4719	0.4953	0.5211	0.5498	0.5818	0.6177	0.6584	0.7048	0.7582	0.8204	0.8937
0.1	0.8994	0.9882	1.097	1.232	1.405	1.634	1.954	2.428	3.206	4.719	8.937

hot face temp = 2732°F cold face temp = 2710°F

$\mathcal{E}_2 = \mathcal{E}_3 = 0.45$ semi pattern length = 0.1 inches

K= 25	0.0211	0.02115	0.02119	0.02124	0.02128	0.02133	0.02137	0.02142	0.02146	0.02151	0.02156
10	0.05113	0.0514	0.05166	0.05193	0.0522	0.05248	0.05275	0.05303	0.05331	0.0536	0.05389
1	0.3502	0.3629	0.3766	0.3913	0.4072	0.4245	0.4433	0.4639	0.4865	0.5113	0.5389
0.1	0.8435	0.9212	1.015	1.129	1.273	1.459	1.708	2.059	2.594	3.502	5.389

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. .... OF ....

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
		DATE	DATE

hot face temp = 2732 °F   cold face temperature = 2730 °F.

$\varepsilon_2 = \varepsilon_3 = 0.85$ semi pattern length = 0.1 inches.

Ratio of Heat Flow Through Porous Material To Heat Flow Through Solid Material

	k=25	20	10	5	1	0.5	0.1	0.01
$P = 1\times10^{-6}$	0.05249	0.06476	0.1216	0.2169	0.5807	0.7347	0.9327	0.9928
0.1	0.05277	0.06518	0.1231	0.2217	0.6165	0.793	1.029	1.102
0.2	0.05305	0.06561	0.1247	0.2267	0.657	0.8613	1.147	1.239
0.3	0.05333	0.06604	0.1263	0.232	0.7032	0.9425	1.295	1.414
0.4	0.05361	0.06648	0.1279	0.2375	0.7564	1.041	1.488	1.647
0.5	0.0539	0.06693	0.1295	0.2433	0.8183	1.161	1.748	1.972
0.6	0.0542	0.06738	0.1312	0.2494	0.8912	1.314	2.118	2.456
0.7	0.05449	0.06784	0.133	0.2557	0.9789	1.513	2.687	3.255
0.8	0.05479	0.06830	0.1348	0.2624	1.085	1.782	3.674	4.826
0.9	0.05509	0.06877	0.1366	0.2695	1.216	2.169	5.807	9.327
1.0	0.0554	0.06925	0.1385	0.277	1.385	2.77	13.85	138.5

HATCH ASSOCIATES

CALCULATION SHEET

SHEET NO. ....OF........

DESCRIPTION	PROJECT NO.	MADE BY	CHECKED BY
		DATE	DATE

hot face temperature = 2732 °F cold face temperature = 2710 °F

$\varepsilon_2 = \varepsilon_3 = 0.85$ Semi pattern length = 0.1 inches.

Ratio of Heat Flow Through Porous Material To Heat Flow Through Solid Material.

P	K = 25	20	10	5	1	0.5	0.1	0.01
1×10^{-6}	0.05202	0.06419	0.1206	0.2153	0.5784	0.7329	0.9321	0.9928
0.1	0.05230	0.06461	0.1221	0.2200	0.6139	0.7909	1.028	1.102
0.2	0.05257	0.06503	0.1236	0.2250	0.6541	0.8588	1.146	1.239
0.3	0.05285	0.06545	0.1252	0.2302	0.6998	0.9395	1.294	1.414
0.4	0.05313	0.06589	0.1268	0.2356	0.7525	1.037	1.486	1.647
0.5	0.05341	0.06632	0.1284	0.2413	0.8137	1.157	1.746	1.971
0.6	0.05370	0.06677	0.1301	0.2473	0.8858	1.308	2.115	2.455
0.7	0.05399	0.06721	0.1318	0.2535	0.9719	1.505	2.682	3.254
0.8	0.05428	0.06767	0.1335	0.2601	1.077	1.772	3.665	4.824
0.9	0.05458	0.06813	0.1353	0.2671	1.206	2.153	5.784	9.321
1.0	0.05488	0.0686	0.1372	0.2744	1.372	2.744	13.72	137.2

$K_{Fe} \approx 15$

K_{sponge} (KIRUNA) ≈ 3.0

HATCH ASSOCIATES	CALCULATION SHEET			SHEET NO. ....OF........
DESCRIPTION Sponge Iron Heat Transfer		PROJECT NO.	MADE BY DATE	CHECKED BY DATE

Take a bed thickness of 0.3m and a pore size of 1×10^{-4} m.

T at the wall = 1500°C = 1773 °K T_{centre} = 800 °C = 1073 K.

ϵ = 0.6 for sponge iron

$$\Delta T / x = \frac{1500-800}{.3m} = \frac{700}{0.3} = 2333.33\ °C/m$$

assuming very little material between pores

ΔT accross a pore = $(1 \times 10^{-4} m)(2333.33) = 0.2334\ °C$

taking the best case $T_1 = 1773 K$ $T_2 = 1773 - .2334 = 1772.77 K$

$\sigma = 5.669 \times 10^{-3}\ W/m^2/°C$ $\frac{Q}{A} = \sigma \epsilon (T_1 - T_2)$ $= 5.69 \times 10^{-8} (.6)(1773^4 - 1772.77^4)$

$= 174.38$

$Q_{RAD} = 174.38/.2334\ °C = 747.11\ W/m^2\ °C$

$k_{effective} = 747.11 \times 1 \times 10^{-4} m = 0.075\ W/m\ °C$.

pure iron	k = 62.3 W/m °C
cast iron	k = 51.9 W/m °C
air at 760°C	k = 0.0677 W/m °C

2

APPENDIX D Chemical Reactivity of Silicon Carbide

The Carborundum Company
Refractories Division
Refractory Materials Plant
P.O. Box 187
Keasbey, New Jersey 08832
Telephone 201/738-4600
Telex: 84-4446
A Kennecott Company

Chemical Resistance of Carborundum Refractories

Chemical resistance unquestionably is the most involved characteristic to evaluate and record of all qualities of refractories upon which a selection is made. Both satisfactory performance or complete unacceptability of a given material may be experienced for certain chemical environments, depending on the variations of the factors that make up the environment. This makes positive and/or complete prediction of inertness to attack virtually impossible. It is obvious that performance data cannot be developed for all likely combinations of variable conditions.

As an elementary principle of refractories selection and use, acid materials will resist acid conditions and basic materials will withstand basic conditions. Our silicon carbide compositions are acid in nature, aluminum oxide is amphoteric (can act as either acid or base), and mullite materials range from acid to neutral. Thus, in general, where severe basic environment exists, other refractories such as chrome or magnesite are indicated. This fundamental relation is affected by a further principle (law) that as temperature increases, the rate of reaction increases between a gas, liquid or molten substance and the refractories in contact therewith.

Temperature certainly is the most critical variable. Its effect on Carborundum refractory materials under a variety of chemical environments is not well documented but is better known than the effects of other variables. The chart which follows will serve only as a general guide to usefulness under elevated temperature conditions. Actual selection of refractories frequently will be made on the basis of performance due to other controlling characteristics, e.g. maximum temperature resistance, thermal shock resistance, etc.

Other seemingly minor variable factors affect performance of refractories in a critical manner. It is known, for example, that when iron is present in a refractory in a significant amount, there is a breakdown of CO to CO_2 and C within the refractory to build up a carbon deposit around iron impurities. This buildup of carbon often creates severe spalling and ultimate refractory failure.

Presence of vanadium in oil feedstock likewise is a factor which affects refractories life/performance significantly. The vanadium oxide reacts with the refractories to form vanadium compounds which melt at lower temperature than normal refractory use temperature. It may be controlled with an additive providing the reaction or a catalyst is not negatively affected thereby.

These two examples are cited to illustrate that "Chemical Resistance", as an important characteristic to be considered in every selection of refractories for the Process Industry, is a complicated subject. There are many, many other variables which may appear of little consequence. As a policy, therefore, it usually is desirable to have customers test refractories under the specific anticipated environment conditions. Whenever appropriate, samples should be offered to customers for testing. The information contained herein, backed by our Application Engineering Department, will be quite helpful in establishing the materials and specific compositions to test.

Carbofrax® and Refrax® Silicon Carbide

Oxidation is a common cause of ultimate failure of refractories containing silicon carbide.

Failure may occur due to attack on the bond material or on the silicon carbide, or simultaneous attack may occur with failure caused mainly by the formation of intergranular silica through one of the following reactions:

$SiC + 2CO \longrightarrow SiO_2 + 3C$

$SiC + 2O_2 \longrightarrow SiO_2 + CO_2$

and $SiC + 3CO_2 \longrightarrow SiO_2 + 4CO$

The resistance to attack by various gaseous environments can, however, be considerably improved by various bonding methods or surface treatments. A number of compositions containing silicon carbide are available, under the general nomenclature of Carbofrax and Refrax, which exhibit varying degrees of resistance to oxidation.

It should also be recognized that temperature considerably influences the rate of reaction of one component with another and there usually exists some temperature level below which the refractory will give adequate and economic service.

The following tables give some indication of the behavior of Carborundum products in various gaseous and chemical environments.

Form A-2587 Effective 3/81
Supersedes C-196
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CARBORUNDUM

Carbofrax® and Refrax® In Gaseous Environments

Environment	Behavior	Comments
Carbon Monoxide (CO)	G	Carbon deposited at 583°C. Slow attack at 1200°C.
Carbon Dioxide (CO_2)	G	Slow attack at 816°C.
Chlorine (Cl_2)	N.R.	Vigorous Reaction at 600°C.
Fluorine (F_2)	N.R.	Destructive at all temperatures.
Hydrogen (H_2)	G	Moderate attack at 816°C. Rapid attack at 1100°C.
Hydrocarbons (Cn Hx)	G	Reaction at 500°C with evolution of H_2 and deposition of C.
Hydrochloric Acid Gas (HCl)	G	No reaction up to 1300°C.
Hydrofluoric Acid Gas (HF)	N.R.	Attack at all temperatures.
Hydrogen Sulphide (H_2S)	F	Moderate attack at 1100°C in the presence of water vapor.
Sulphur (S)	F	Reacts at 800°C.
Sulphur Dioxide (SO_2)	G	Up to 11% concentration with Sulphur absent. In excess Oxygen no reaction up to 1320°C. Accompanied by S and CO, reacts at 1380°C.
Sulphur Trioxide (SO_3)	G	Attack may occur if free Sulphur present. See Sulphur and Sulphur Dioxide.
Ammonia (NH_3)	G	Attack may occur if Hydrogen present. See Hydrogen.
Bromine (Br_2)	F	
Air	G	
Steam (H_2O)	F - G	Oxidation at 800°C - 1200°C.
Nitrogen (N_2)	G	
Phosphorous Pentoxide (P_2O_5)	N.R.	Reacts 700°C.

G = Good. F = Fair. N.R. = Not Recommended.

Alfrax® and Mullfrax® In Gaseous Environments

Alfrax, and Mullfrax behave well in the following atmospheres:

Carbon Monoxide	(CO)
Carbon Dioxide	(CO_2)
Steam	(H_2O)
Bromine	(Br_2)
Hydrochloric Acid Gas	(HCl)
Sulphur Dioxide	(SO_2)
Sulphur Trioxide	(SO_3)
Sulphur	(S)
Ammonia	(NH_3)
Hydrogen Sulphide	(H_2S)
Hydrocarbons	(Cn Hx)
Air	

Alfrax and Mullfrax show fair resistance to attack by:
Chlorine

90% Al_2O_3, and above, Alfrax materials offer good resistance to attack by:
Hydrogen and
Hydrofluoric Acid

All Alfrax and Mullfrax are attacked by:
Fluorine

Guide to the Erosion and Corrosion Resistance of Refractories (continued)

Substance	M.P.°C	Recommended Carborundum Refractories	Application	Comments	Other Refractories Gener ılly Used
Chromium and (Ferro Chrome) (Cr)	1830	B1 S3			B11, B13
Chromium Trioxide (Cr_2O_3)	1990	A1, A2			B11, B13
Chromium ($CrCl_3$)	Sublimes 1300	A1, A2			B11, B13
Cobalt (Co)	1480	N1, N2, N3, N4			N13, N14, N15
Cobalt Oxide (Co_3O_4)	1300	A1		A1 to 1150°C Reaction at 1590°C	
Copper (Cu)	1080	A1, A2 N1, N2, N3, N4 S2	Roasting Smelting Converting Refining	S4 attacked at 1150°C Tendency of basic hydration — wet charge	B11, B13, B16 N13, N14
Cuprous Oxide (Cu_2O)	1235	N1, N2, N3, N4 S2		Cu_2O/S_1O_2 eutectic 1060°C	B11, B13, B16
Cryolite ($Al F_3$ 3NaF)	1000	A1, A2 S2, S3, S4	Reduction Cells Melting		S11
Ferrous Sulphide ($Fe S_2$)	1193				B11, B13 S11
Glass (Opal, Soda-Lime, Borosilicate)	1500 approx.	N1	A18 Roofs A19) Glass N1) Contact N14 Walls, Ports, Burner Blocks B12 Ports Checkers A11, B11, B12 B15 Checkers		A11, A18, A19 B11, B12, B15 N14
Gold (Au)	1063	N3, N4 S3			N13 S11
Iron and Steel	1150 to 1535 pure — grey — white	N1, N3, N4 S3	Iron and Steel Production	Acidic and Basic processes Fe + 4%C + attacks SiC only slowly	A11, A12, A14, A18 B13, B14, B16 N13, N14 S11
Iron Oxide (Fe_2O_3)	1565	A1, A2 N1, N3, N4 S3		A1, A2, S1 to 1100°C	As for Iron
Lead (Pb)	330	A1, A2 N1, N2, N3, N4, N5			A11, A12 B13, B14, B15, B16 S11
Lead Monoxide (PbO)	888	N1, N2, N3		S4 not attacked by $PbO:SiO_2$ at 900°C	A11, A12 B11, B13, B14 N14, N15
Lead Sulphide (PbS)	1114		As for Lead Monoxide		
Lead Fluoride (PbF_2)	855	N1, N2			B11, B13 S11
Magnesium (Mg)	650	A1, A2 N1, N2, N3, N4 S2, S4	Electrolytic Process S4 slightly attacked		A11, A17 B11 N1

HATCH ASSOCIATES	CALCULATION SHEET	SHEET NO. . . . . . . OF . . . . . .

DESCRIPTION	PROJECT NO.	MADE BY DATE	CHECKED BY DATE
Muffle Furnace Experiments			

Nov. 10, 1987. Experiment #1

The tip of the thermocouple was located 10cm from the hot face. After approximately 80 minutes the rate of temperature increase at the thermocouple decreases. The reason given is that heat losses are occuring at the end of the SiC tube. However the tube wall would be hotter than the material in the bed and thus heat would not flow from the ore/coal mix to the tube walls and out of the system.

It is interesting to note that at approximately the same time as the temperatur starts to level out, heavy smoke started to flow from the cold end. The experimente state that the depth of material completely metallised over a period of 115 minutes was 8cm or an effective rate of .7mm/minute. Our examination of the material indicate that there was still a considerable amount of coal present in the material that was supposed to be 100% metallised.

Experiment #2

In this experiment the temperature measured by the thermocouple once agai started to level out after 60-70 minutes. The temperature at which phenomeno started to occur was once again in the 450-500°C range. Thus it is likely that factors other than heat losses were responsible. The ore/coal mixture was metallised to a depth of 4.5cm over a period of 90 minutes. This corresponds to a rate of $^{4.5cm}/_{90\,minutes}$ = 0.05cm/min = .5mm/min.

It should be noted that in both of these experiments heat transfer occurred by radiation to the ore/coal mixture exposed at the uninsulated end of the tube. It is likely that as the CO left the tube at this end it burne and produced local temperatures much in excess of the furnace temperat which was in the range 1000-1400°C. This is not likely to be representative o the heat transfer through the tube to the material though it is a good representation of the heat transfer occuring ~~at~~ the end of the reactor tub in the bench-scale process.

	CALCULATION SHEET			SHEET NO. ....OF........
DESCRIPTION		PROJECT NO.	MADE BY DATE	CHECKED BY DATE

Experiment #3

This experiment was carried out in a tube placed on its end with a cover on the top of the tube. Thus there was no direct radiative heat transfer to t coal/ore mixture. The investigators found that the material was metallized to a depth of 2.0 cm over a period of 25 minutes. This gives an effective react rate of 0.8 cm/min which is even faster than for the case when the material is directly exposed (experiments 1 and 2). This is rather unlikely since exposed material would received heat by radiation and also would receive the added benefit of the CO combustion.

Experiment #4.

This experiment was carried out in the same manner as experiment #3 except that the tube was flipped on its side after 25 minutes in order to expose the ends for the last 10 minutes of heating. The ~~top~~ material at the top of th tube metallized to a depth of ~1.0 cm while that on the bottom of the tube (which had received ~~not~~ heat for the first 25 minutes) was metallised to depth of ~0.5 cm. It can be concluded then that the rate of reaction was ~0.5 mm/min for the last 10 minutes when heat transfer was due radiation. Once again CO combustion would give considerable benefit. Based on this rate of reaction over the last 10 minutes, the rate of reaction for the first 25 minutes be obtained by difference (ie depth reacted at top face - depth reacted at bottom face)/1 Thus an effective reaction rate of 0.2 mm/min is obtained for the first 25 min of heating.

November 12, 1987
W-K. Lu

EXPERIMENTS IN A MUFFLE FURNACE

(a) Seven experiments were carried out in October 1987. Experimental results have been presented and reacted specimens shown at the meeting on October 26, 1987 in Toronto. Iron ore used in these experiments was magnetite concentrate from Northern Ontario.

(b) Due to the fact that almost all Canadian iron ores are produced on the north shore of the St. Lawrence and are hematite, fines from Wabush mines were used in experiments carried out on November 10 and 11, 1987. Properties of ore and coal are documented on the following page.

(c) November 10, 1987 Experiments: Growth of metal shell under very unfavourable conditions.

A one-dimensional reacting system was placed at the spot where the door used to be. Crucibles were wrapped heavily in insulation materials, except one end which faced the chamber of the furnace. Details are shown on the enclosed figures. To the best of our estimation, the depth of mixtures which are completely metallized is at least 8 cm in Experiment #1 and 4.5 cm in Experiment #2.

(d) November 11, 1987 Experiments: Growth of metal shell and the effect of direct radiation.

Details of Experiments #3 and #4 are shown on the enclosed figures. Taking note of the fact that it takes ten minutes for heating up the crucible to initiate the reactions, the depth of the metal shell is about 20 mm in 25 minutes in Experiment #3. Under otherwise the same experimental conditions, at minute 25 in Experiment #4 the cover was removed and the crucible turned 90°, direct radiation through both ends results in a much higher degree of metallization of the specimen. Even though, the range of temperature in Experiments #3 and #4 is rather on the low side.

RAW MATERIALS USED IN EXPERIMENTS CONDUCTED

NOVEMBER 10 and 11, 1987

(1) Wabush Fine, Received from Wabush Mines in November 1985

Chemical Analysis 65.26% Fe
2.5% SiO_2
1.83% Mn

Size Analysis

+ 20 mesh (850 m)	0 wt.%
- 20 mesh + 30 mesh (600 m)	9.3 "
- 30 mesh + 50 mesh (300 m)	26.1 "
- 50 mesh + 100 mesh (150 m)	29.4 "
- 100 mesh	35.2 "

It was used as received.

(2) High Volatile Coal, Received from Dofasco in 1987

Volatile Matter 30.6%

Ash in the range of 6 to 7%

Size Analysis

	As-Received	Used in Experiment
+ 20 mesh	45.8 wt.%	0 wt.%
- 20 mesh + 30 mesh	8.1 "	11.9 "
- 30 mesh + 50 mesh	15.7 "	23.2 "
- 50 mesh + 100 mesh	12.5 "	18.4 "
- 100 mesh	17.9 "	46.4 "

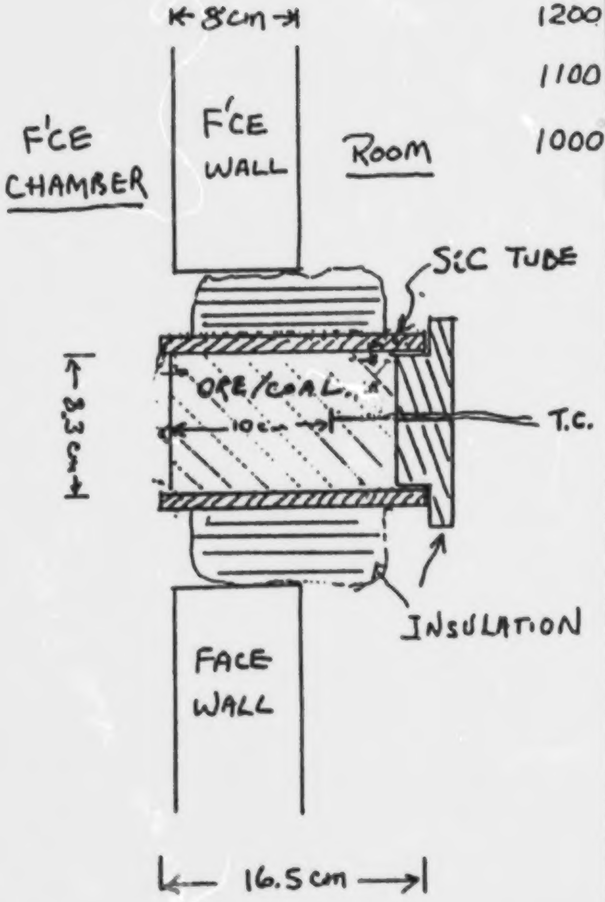
About 50% of coal needs grinding.

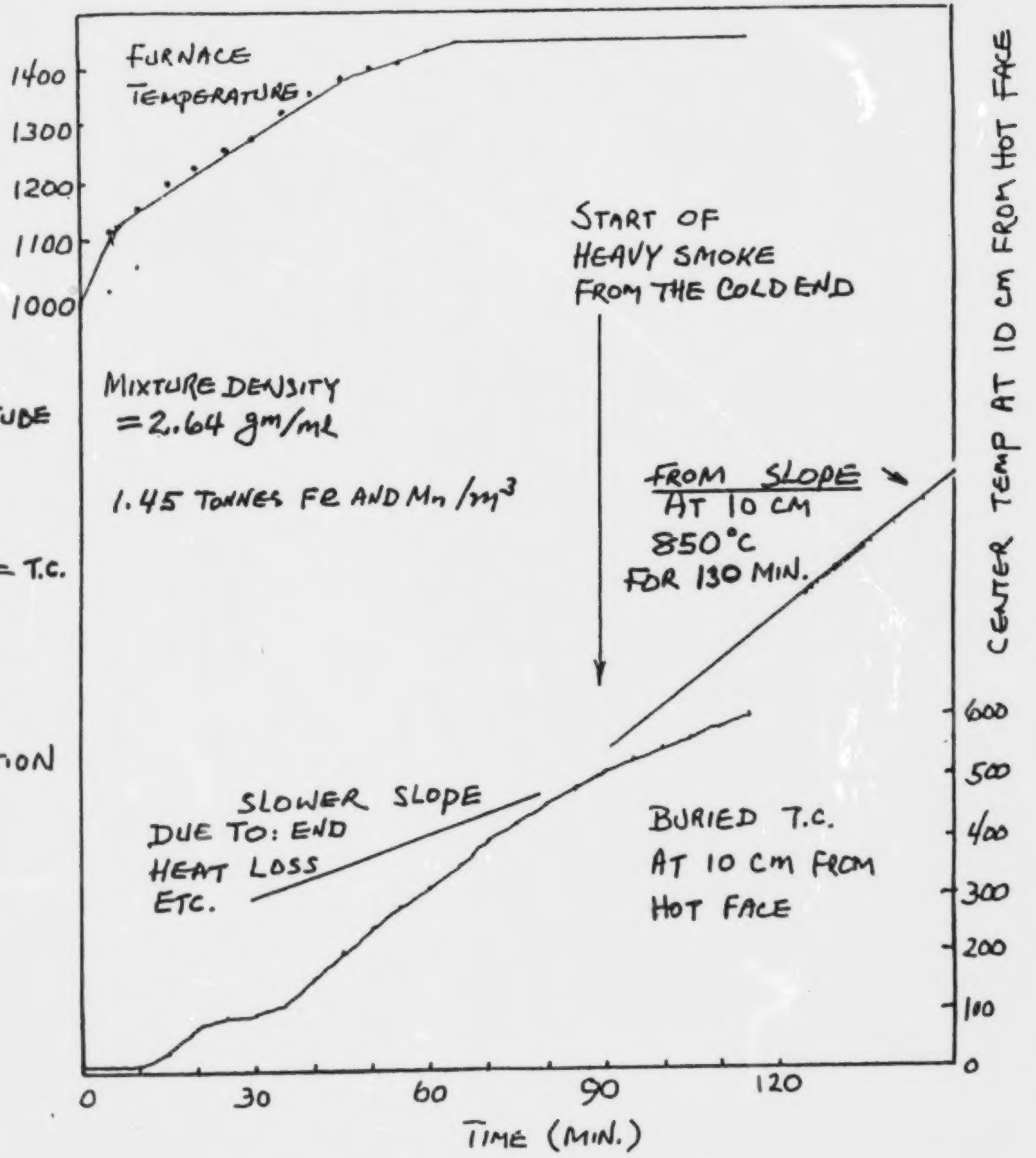
(3) Coal/ore mixture has 18% coal and 82% ore, by weight.

NOTE: Three high volatile coals have been used. They were supplied by Dofasco Inc., Stelco Inc. and Ontario Hydro (thermal coal). No difference was detected in our experiments.

NOVEMBER 10, 1987

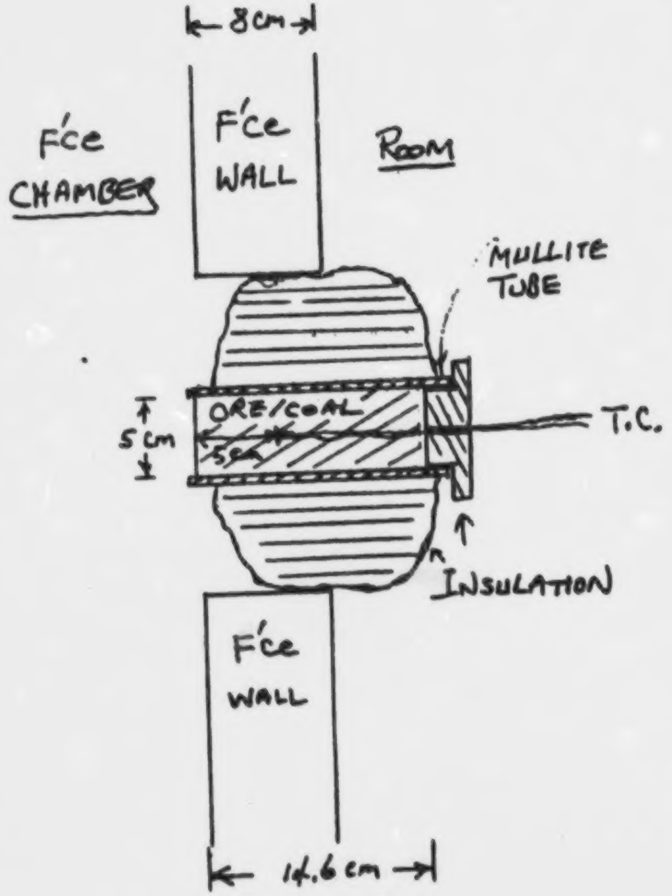
EXP. NO. 1


F'CE CHAMBER
← 8cm →
F'CE WALL
ROOM
SiC TUBE
ORE/COAL
10cm
T.C.
← 8.3cm →
INSULATION
FACE WALL
← 16.5cm →

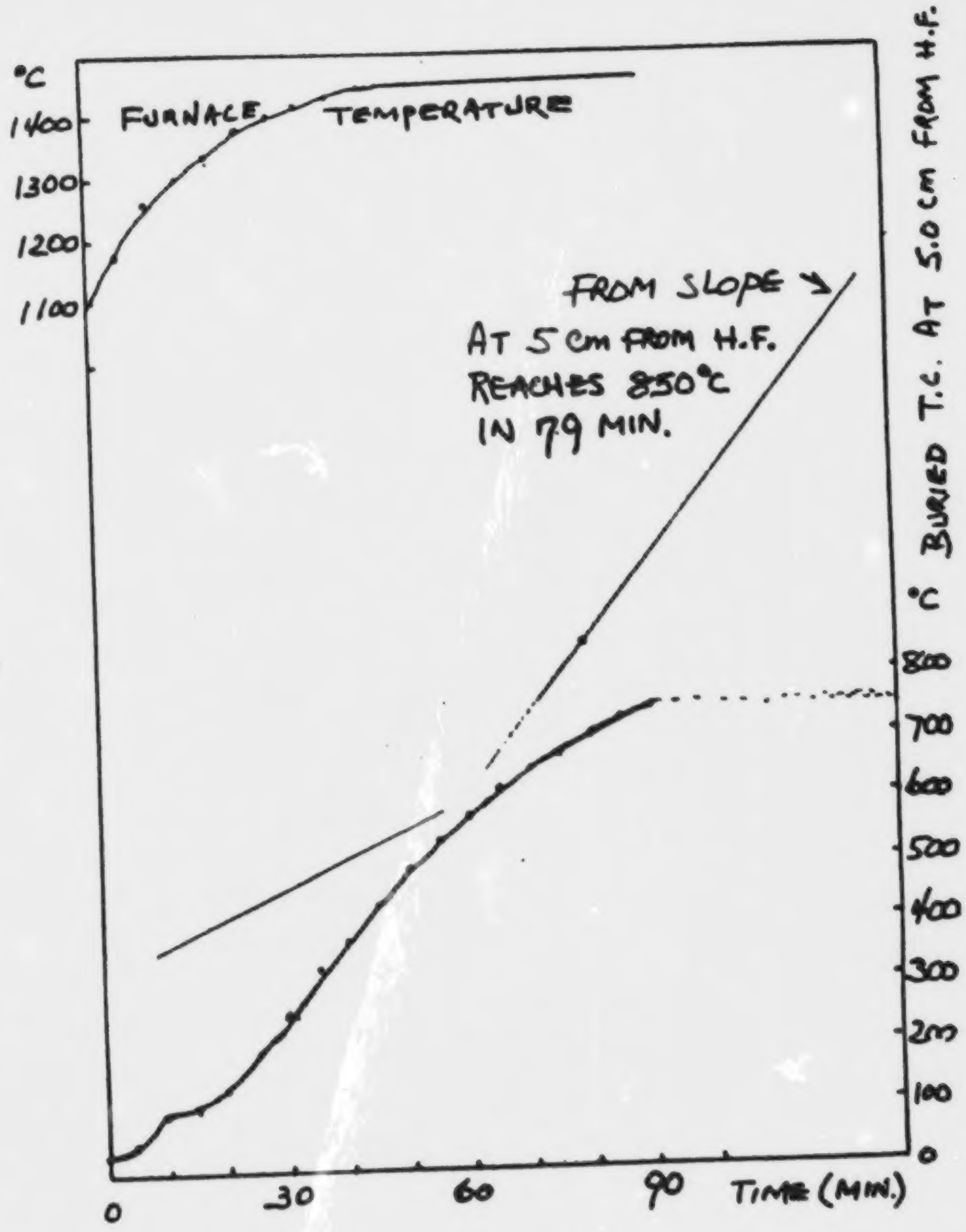


NOVEMBER 10, 1987

EXP. NO. 2

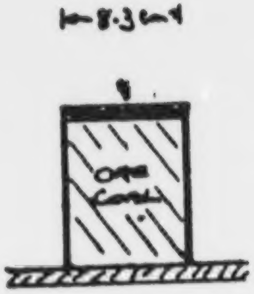


F'ce CHAMBER
← 8cm →
F'ce WALL
ROOM
MULLITE TUBE
ORE/COAL
5cm
5cm
T.C.
INSULATION
F'ce WALL
← 14.6cm →



AT 5 CM FROM H.F. REACHES 850°C IN 79 MIN.

NOVEMBER 11, 1987


8.3cm
ORE
COAL

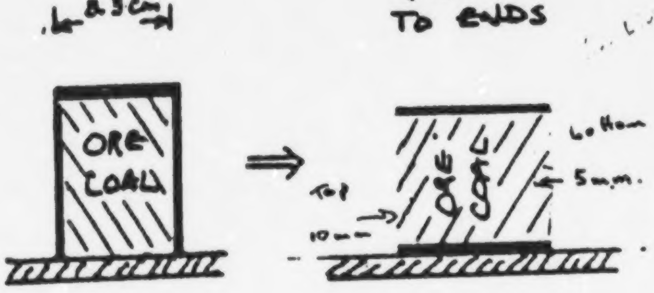
~20cm over 25 minutes.

Exp. #3 . MIN. 0 – 35

10 min before reaction starts.

Exp #4

DIRECT
RADIATION
TO ENDS


8.3cm
ORE
COAL
⇒
Top
10mm
bottom
5mm.
ORE
COAL

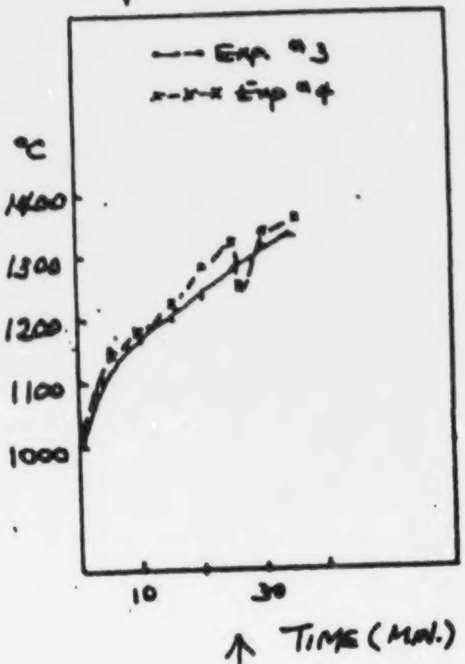
MIN. 0 -25

MIN. 26 -35

radiation at ~ 1300 °C

~10 minutes.

BLACK SMOKE STARTS
(CARBONIZATION)
MIXTURES REACH 500-600°C



AT MIN. 20
F'ce DOOR OPENED
TOP COVER REMOVED
CRUCIBLE TURNED 90°

PROJECT REPORT

HATCH ASSOCIATES LTD. CONSULTING ENGINEERS

HATCH
ASSOCIATES

P.R. #5006.Tor.12
File #5006.Tor.3

STELCO - WORLD STEEL TRENDS

December 21, 1987

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1.0 INTRODUCTION

The overall study is to determine the technical and economic potential for the LB Process for the production of virgin iron units, leading to a possible decision for the support for a pilot plant - a major financial commitment. This particular report reviews and discusses the world steel markets - consumption, total and net trade, production and means of production. No attempt has been made to present a comprehensive picture of price or price forecasts. An objective is to identify the trends and future production requirements and assess the possibility for the LB Process finding a place in future steelmaking decisions.

Complex as the total industry of iron and steel is, it is useful to recall the basic movement of raw materials from source to consumer in simplified terms.

Virgin Iron Units:
The production of this material, from raw resources of iron ore, energy and carbon, plus chemical/reaction balance materials such as limestone, is the starting point for steel. Virgin iron units are primarily produced as blast furnace (BF) hot metal (worldwide production of 483 metric million tons in 1985). Some of the iron units, as pig iron, are used by iron foundries in combination with steel scrap. A small extra quantity of solid virgin iron units is produced by direct reduction (at a level of 11 million metric tons in 1985). This production of Direct Reducted Iron (DRI) falls far short of the optimistic forecasts which had been made in the 1970's.

Scrap Steel:
This material is both a steel industry by-product and a feed material to the steelmaking process.

Steelmaking:
The dominant process is the basic oxygen process (BOF) which includes the well established BOF process and other converted oxygen steelmaking variants. This process is charged with virgin iron units (primarily from the BF as liquid) and up to 30% scrap steel.

The other major process is the Electric Arc Furnace (EAF) charged with scrap steel. In some situations, a varying percentage of the charge consists of DRI when special physical properties are required or DRI production economics are favourable.

Steel Products Shipped:
Of the liquid steel produced, approximately 83-85% leaves the steel plant as marketable product.

The choice of steelmaking process depends upon a number of factors which will be discussed. Of prime importance is the supply of virgin iron units. At present, BF hot metal and DRI are the only economic options for these primary units. The molten product from the LB Process can be considered as a third option, either as a replacement for blast furnace iron or as a portion of the charge make-up to the EAF. As solid pig, a potential market exists as a premium cupola charge material for the production of nodular cast iron.

It is acknowledged that a reduction process which separates gangue from iron at a low energy cost and requires a relatively low capital cost investment would be desirable for steel works, particularly when limited production is required. The regions in the world where this process would hold the most promise for commercialization are identified.

2.0 STEEL CONSUMPTION

All consumption figures are expressed as million tonnes crude steel (first solid ingot) although actual consumption is by finished product.

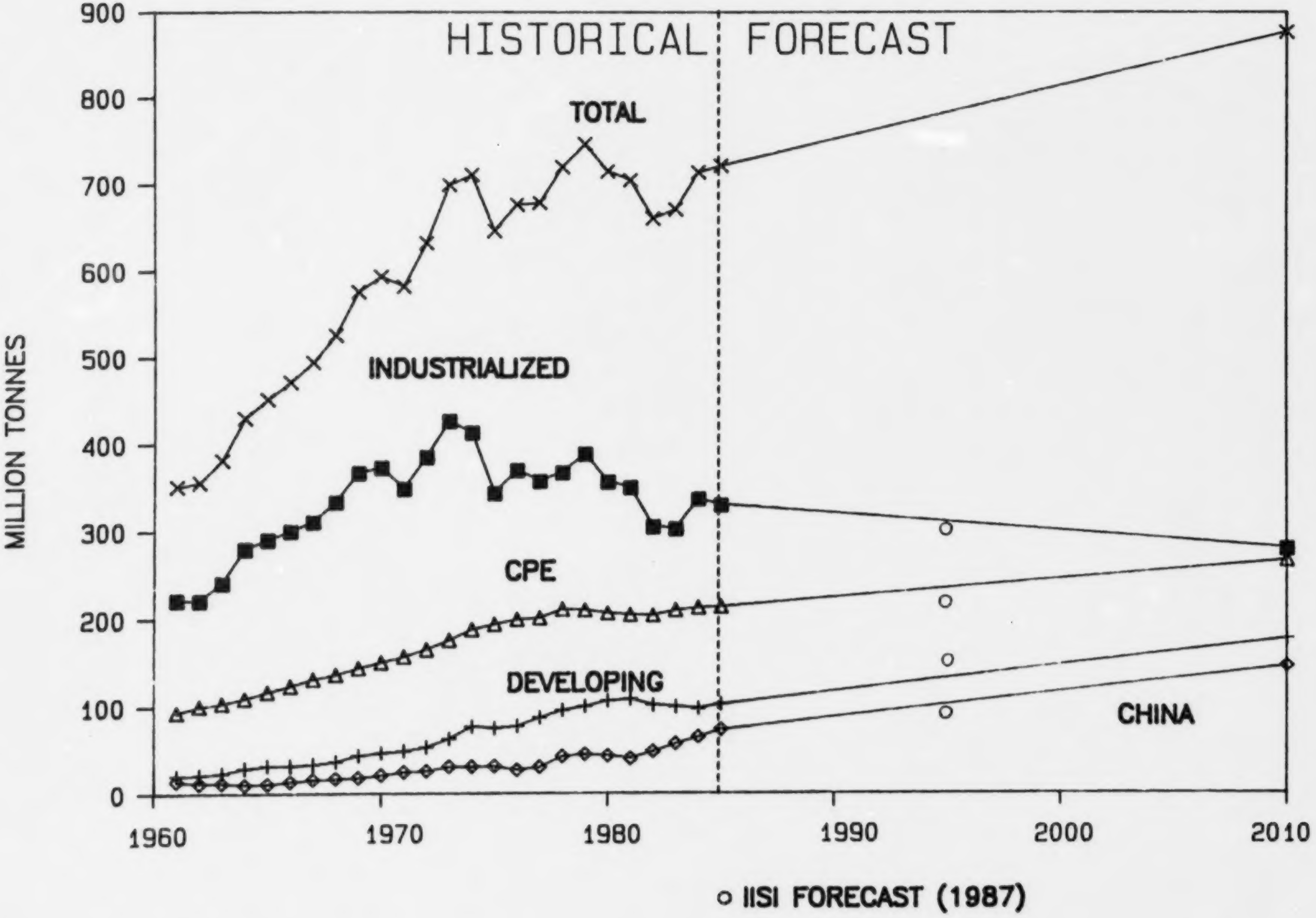
2.1 Historic Trends

From 1960 to 1972 world steel consumption doubled from approximately 350 million tonnes to 700 million tonnes. (Graph 1) In the next 15 years growth became more random and slowed considerably (1986 consumption was marginally greater than 1972). During the latter period the growth rate, or lack of it, varied from region to region. In part this can be explained by the decline in the steel intensity of use as the economics of the industrialized countries mature. The Centrally Planned Economies, (CPE) and developing countries, however, have continued on a consumption growth pattern.

When a country first starts to industrialize, investments are required for infrastructure such as bridges, railways, ports, power stations, etc., which are steel intensive, particularly for the non flat steel products. Steel products such as rebar and light structural steel are at the low end of the value added scale. At a later stage there is a shift to heavy industry, consumer durables, non-durable consumer goods and ship building which require flat products. For countries with oil and gas exploration and extraction activities there has been a need for pipe and tubing products, many of which are produced from flat rolled products. Overall, as a country develops, there is a trend to higher percentage consumption of flat-rolled products (see Appendix 3 as one example).

As production yields have improved, primarily because of the adoption of continuous casting, less crude steel is needed per ton of finished product. Hence crude steel can be a somewhat misleading measure of the industry's activity. Furthermore, thinner and stronger steels often replace thicker steels for the same application with a resultant reduction in demand.

WORLD APPARENT STEEL CONSUMPTION



GRAPH 1

The share of the world's raw steel consumption has shifted in the last 20 years. As shown in Table 1, the industrialized countries' percentage has declined, whereas the developing countries' share has increased threefold.

TABLE 1

World Steel Consumption

	1970		1975		1980		1985	
	%	million tonnes	%	million tonnes	%	million tonnes	%	million tonnes
Industrialized	(63.1)	375.1	(53.4)	346.0	(50.1)	358.5	(46.7)	332.3
Developing	(7.9)	46.9	(11.6)	75.2	(14.9)	106.8	(13.5)	97.2
CPE*	(25.5)	151.7	(30.1)	195.1	(29.0)	207.5	(29.4)	211.3
China	(3.5)	21.0	(4.8)	31.1	(6.0)	43.1	(11.0)	79.0
TOTAL		**594.6**		**647.4**		**715.9**		**719.8**

*USSR, Eastern Europe (known as centrally planned economics or CPE)

Source: SRI Table "World Steel Comsumption"

Industrialized countries experiencing a decline in steel consumption include:

- United States
- Belgium
- Luxemburg
- West Germany
- France
- The Netherlands
- United Kingdom
- Sweden

Countries where steel intensity decline is expected include Canada, Italy, South Africa and Japan(1).

---

(1) Source - Associated Press Interview WEFA Group.

Future Trends

The expected trend in world steel consumption in total and by regions (see Appendix 1) is shown in Graph 1. It is not based on detailed country by country analysis but is rather a qualitative assessment of the probable consumption. It assumes that there will not be any major change of the consumption patterns established for each region since 1972.

Industrialized Countries

Total steel consumption in the industrialized countries is expected to decrease slightly, reflecting reduced domestic demand in North America, Japan and the European Community.

Consumption of steel in the United States and Canada has been adversely affected by downsizing of the automobile. (This has not been a major factor in the case of European countries where small, fuel-efficient cars have been standard for some time.) Manufacturing equipment is shifting to less steel intensive processes. Substitution of competitive materials has made great inroads, such as aluminium, which has replaced steel in much of the container industry.

Reduced Japanese domestic production likely will reflect manufacturing operations moving to other countries. Increased competition in steel intensive industries, such as ship building, can be expected from countries reclassified as newly industrialized, such as Korea, Taiwan, Brazil, etc.

In the European Community the requirement for finished steel products will tend to decrease slightly reflecting the impact of a declining steel intensity in a maturing economy, (growth of per capita steel consumption slower than the per capita G.N.P.), substitution of other materials and shift to higher strength steels.

Developing Countries

The developing countries likely will continue to consume more steel as infrastructure is built and industrialization progresses. Servicing a heavy debt load, however, may be a serious drag on expansion plans for many developing countries. Steel consumption could also be seriously affected by the tension in the Middle East which will have a negative influence on growth. On the positive side, the buoyant business climate in Asia, in particular South Korea, Taiwan, etc., is expected to continue.

Steel consumption by developing regions for 1985, 1990 and 1995 is shown in Table 2.

TABLE 2

Apparent Steel Consumption - Developing Countries
(million tonnes)

Region	1985	1990	1995
Asia	47	60	68
Latin America	28	35	44
Africa	10	11	12
Middle East	16	12	13
	101	118	137

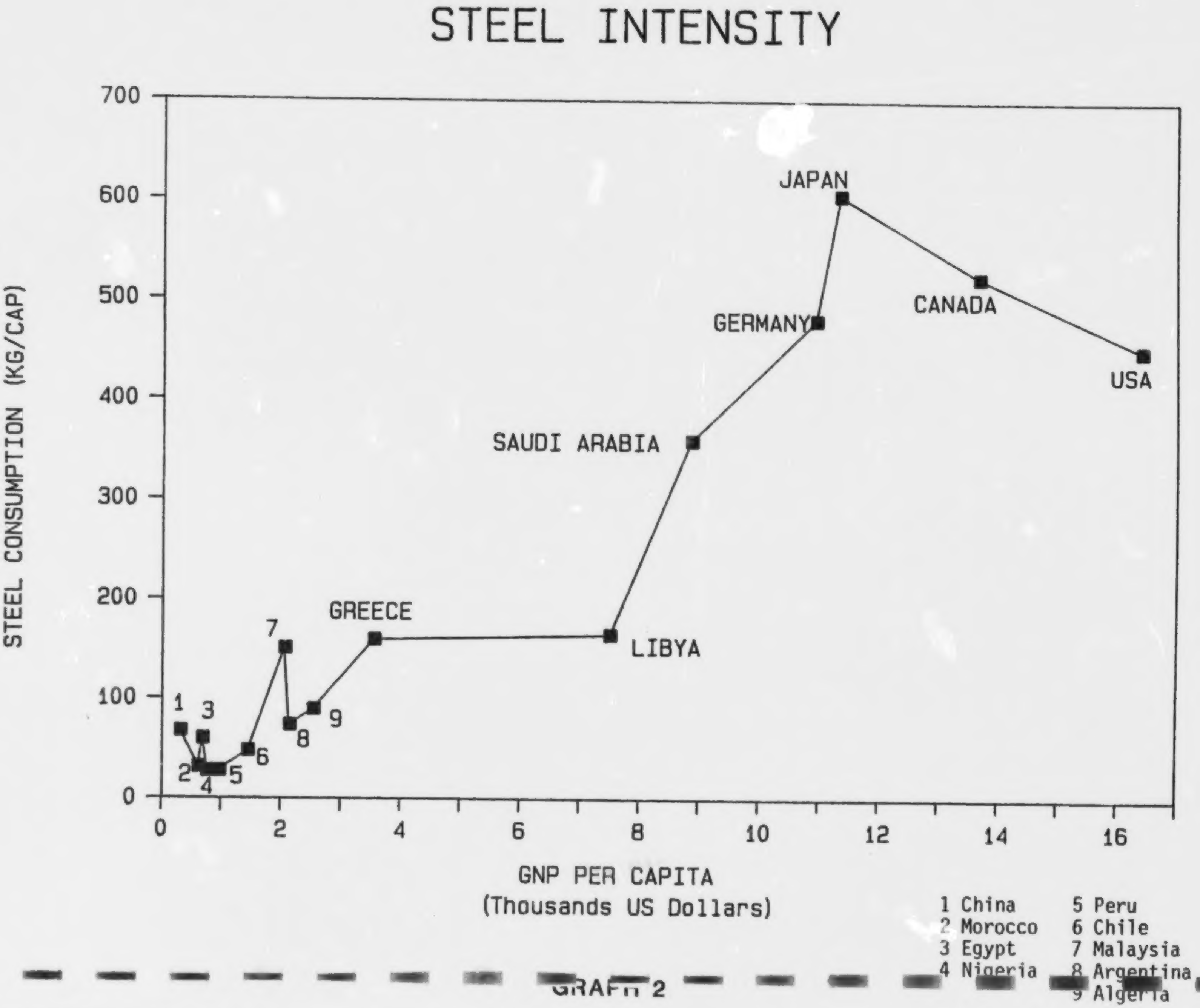
Source - IISI

Centrally Planned Economies

Information developed by the IISI indicates that the steady growth for the CPE, will continue.

China

There is a expectation that China's requirement for steel will be quite extensive. However, this optimistic forecast must be tempered by the difficulty of sustaining high growth rates over a long term. Currently (1985 and 1986) China has a higher per capita steel consumption compared to its GNP than most other developing countries, as shown in Graph 2 (see also Appendix 8).


STEEL INTENSITY
STEEL CONSUMPTION (KG/CAP)
0
100
200
300
400
500
600
700
GNP PER CAPITA
(Thousands US Dollars)
0
2
4
6
8
10
12
14
16
1
2
3
4
5
6
7
8
9
GREECE
LIBYA
SAUDI ARABIA
GERMANY
JAPAN
CANADA
USA
1 China
2 Morocco
3 Egypt
4 Nigeria
5 Peru
6 Chile
7 Malaysia
8 Argentina
9 Algeria

GRAPH 2

3.0 WORLD TRADE

Historically, as a basic industry, steel production has been primarily domestic. However, trade has always occurred with most going from established industrial to emerging countries. Over the past 35 years world trade in steel has increased significantly from about 11% in 1950 to approximately 30% in 1985, i.e. at a rate of about ½% per year. (See Appendix 2). This trend, plus factors such as international iron sourcing, which resulted in iron ore becoming a widely traded world commodity after World War II, are changing the steel business to be much more like other metal commodity businesses, with total world sourcing and competitive world pricing structures.

The major exporters over the past twenty years have been the industrialized countries of Europe, Japan and North America. In the past five years two significant changes have occurred. The United States has become a major net importer and the total imports by China have increased dramatically.

More recently some countries which traditionally have not been considered as exporters of consequence have developed their steel industry to a point where exports are significant. These countries include:

TABLE 3

Steel Exports
(million tonnes)

	1973	1979	1986
Korea	.96	3.0	5.6
Taiwan	.25	1.0	1.9
Brazil	.43	1.39	7.1
Spain	1.72	4.10	7.9
South Africa	.63	2.60	2.9

TABLE 3

Steel Exports
(million tonnes)

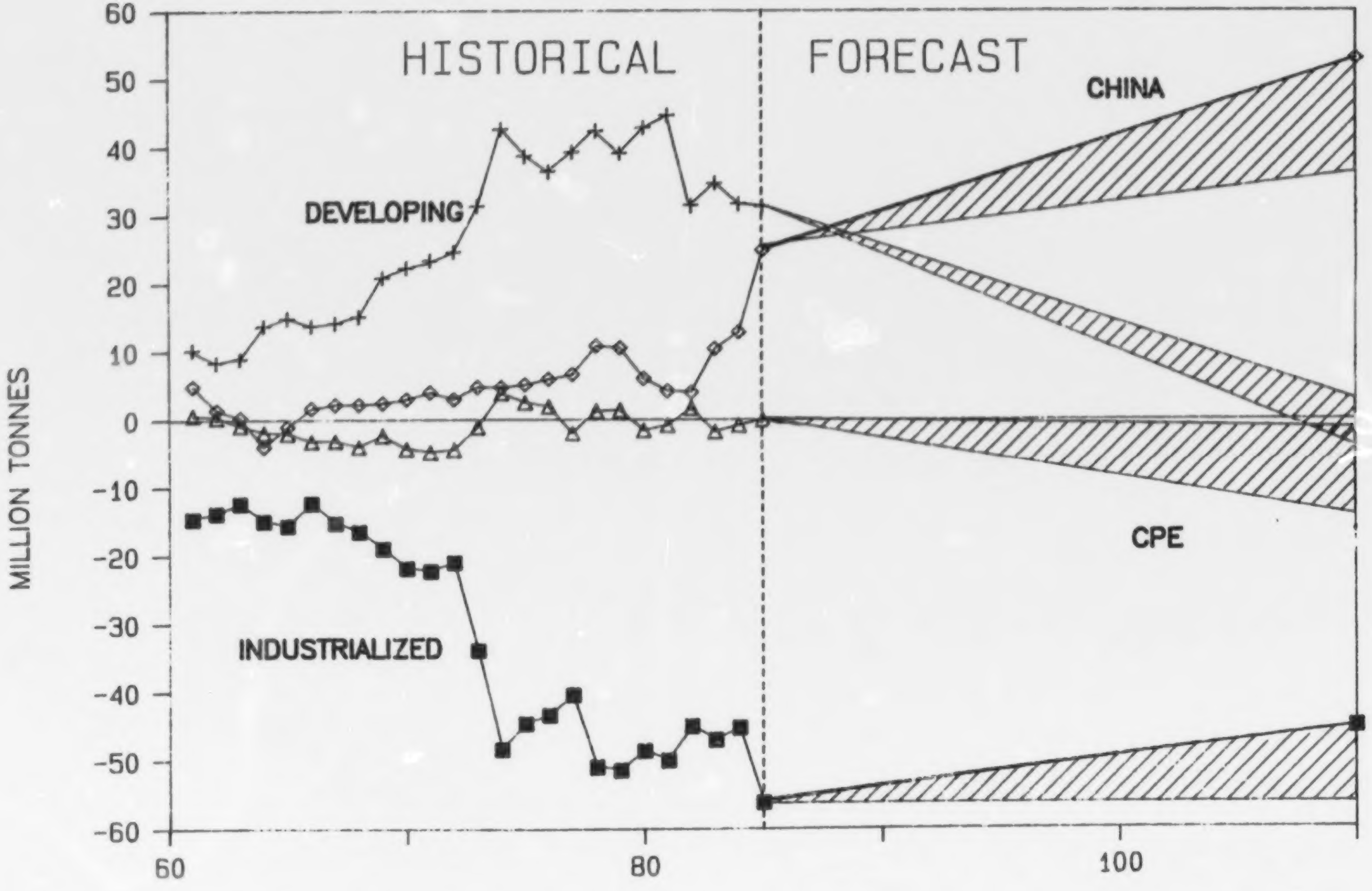
	1973	1979	1986
Korea	.96	3.0	5.6
Taiwan	.25	1.0	1.9
Brazil	.43	1.39	7.1
Spain	1.72	4.10	7.9
South Africa	.63	2.60	2.9

A guesstimate has been attempted for the expected trend in net steel imports, Graph 3.

The following forecasts have been made:

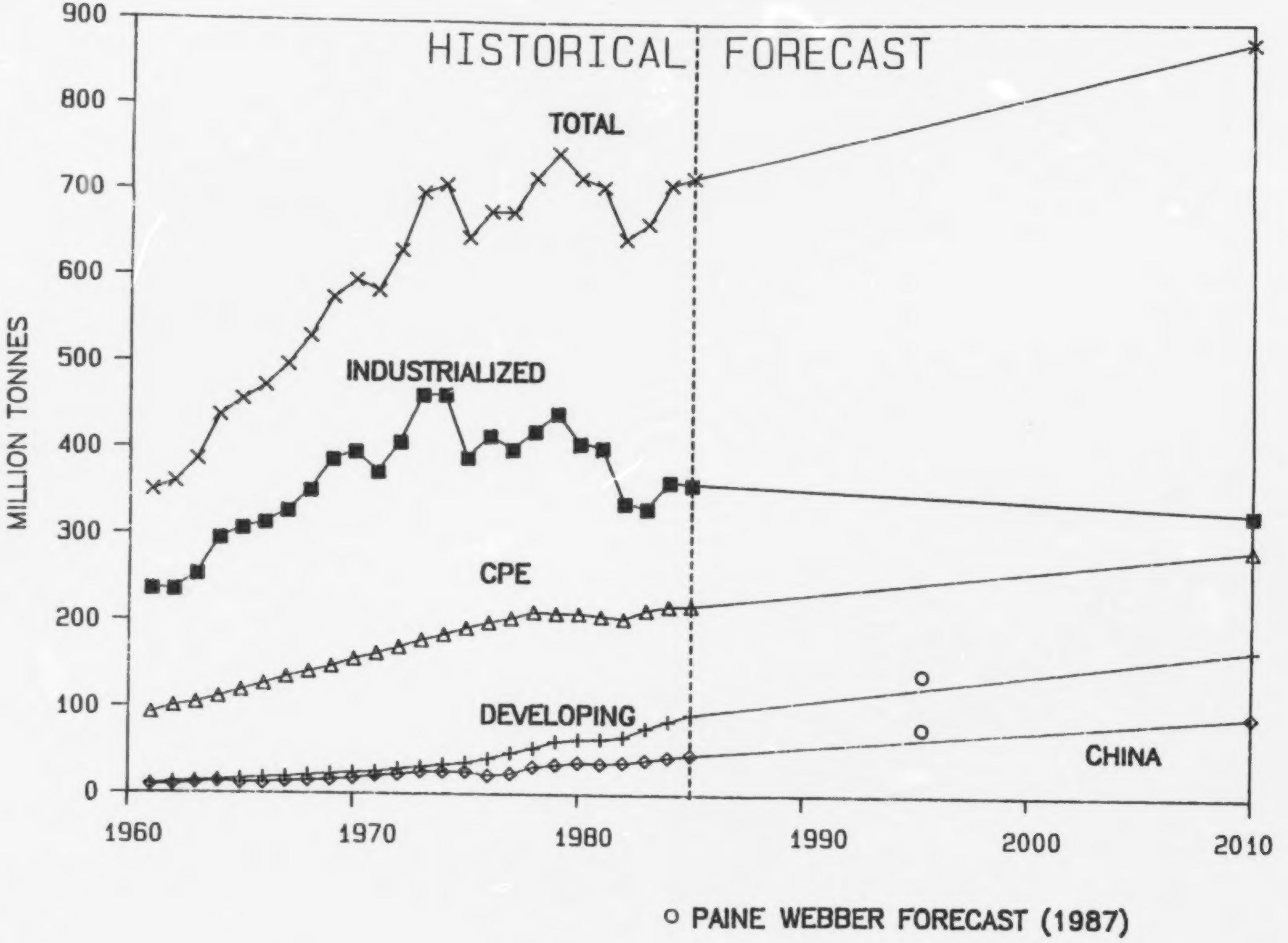
- No major change in imports or exports by CPE countries.
- Continued imports by China at an increasing rate.
- The United States will restrain imports at about the current level of approximately 20 million tonnes.
- Japan will continue to be a major exporter but rate of growth will be moderate.
- The total for all industrialized countries will be about the same in 2010 as in the late seventies and early eighties.
- Net imports in the developing countries will decrease, primarily because of the emergence of the New Industrialized Countries (NIC's), such as Korea and Taiwan, from the ranks of the developing countries.

NET STEEL IMPORTS



GRAPH 3

WORLD STEEL PRODUCTION



GRAPH 4

4.0 STEEL PRODUCTION

Production of raw steel by regions (in million tonnes) is shown in Graph 4. The trend line is identical to consumption for the world. However, the trends for other regions have been modified slightly from the consumption trend line to reflect positive exports by the industrialized countries, and net imports by the developing regions and China. It is assumed the CPE countries will be a zero net importer.

The production of steel by regions (industrialized countries, major developing countries, Eastern Europe and USSR) by process is shown in Graphs 5, 6 and 7.

Industrial Countries

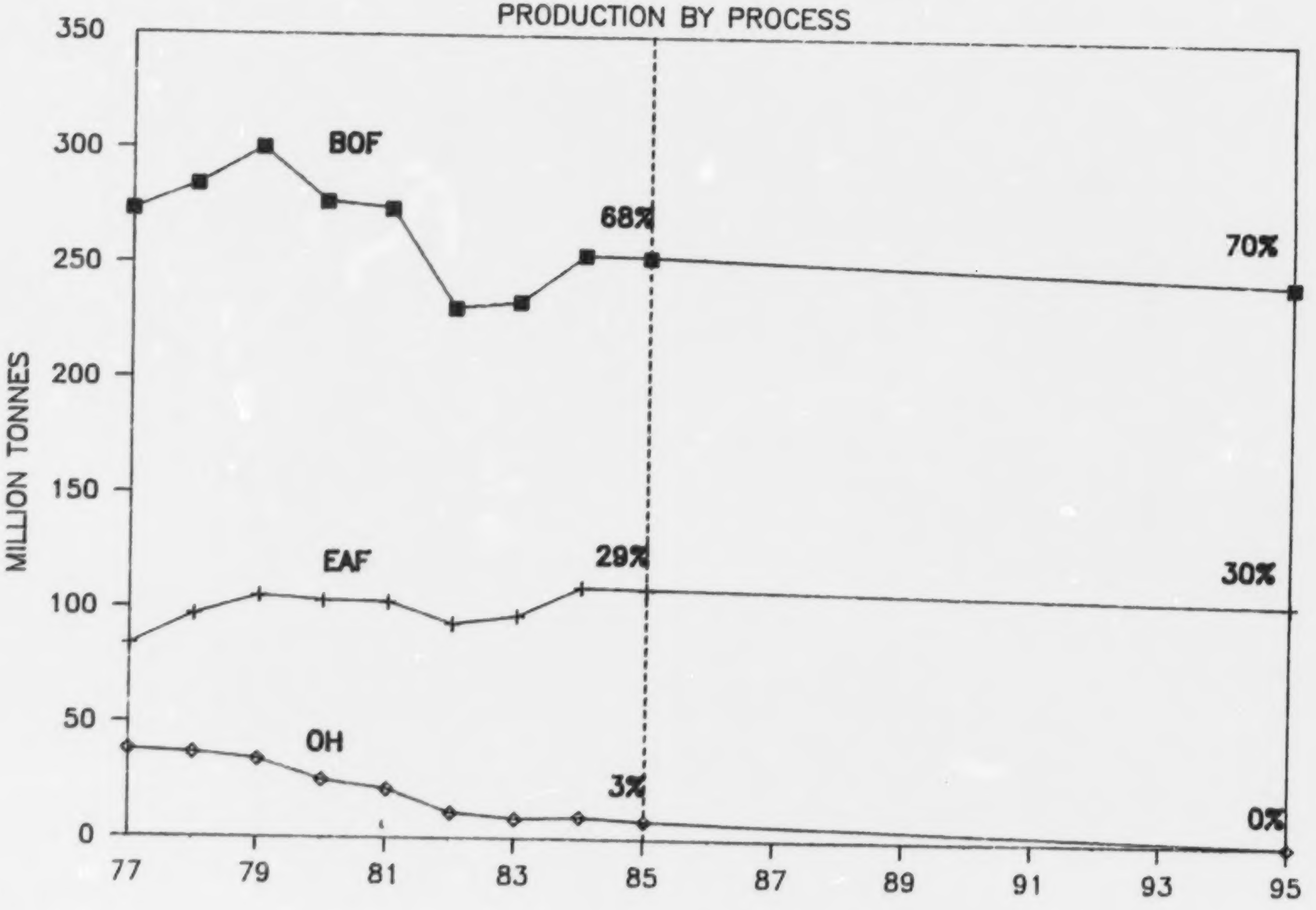
In the industrial countries the BF/BOF route has been the dominant process accounting for almost 70% of total production in the mid eighties. No major change is expected in this percentage for the next 10 years.

Inefficient blast furnaces will continue to be shut-down in favour of more efficient units. The Federal Republic of Germany, for example, with only 40 blast furnaces in 1985, produced more hot metal than 129 furnaces in 1960. The United States is also taking BF capacity out of service and upgrading existing units to meet present and future hot metal requirements.

EAF steel production (see Appendix 4 for list of installations) varies considerably from country to country depending upon the availability and price of electrical energy and to a lesser extent scrap steel. For most industrialized countries EAF account for about 30% of total steel output, but in Spain and Italy, it is more than half as shown in Table 4. Electric furnace production in total for industrialized countries however is not expected to change significantly as the increase in the USA will be offset by a decline in Japan.

INDUSTRIAL COUNTRIES

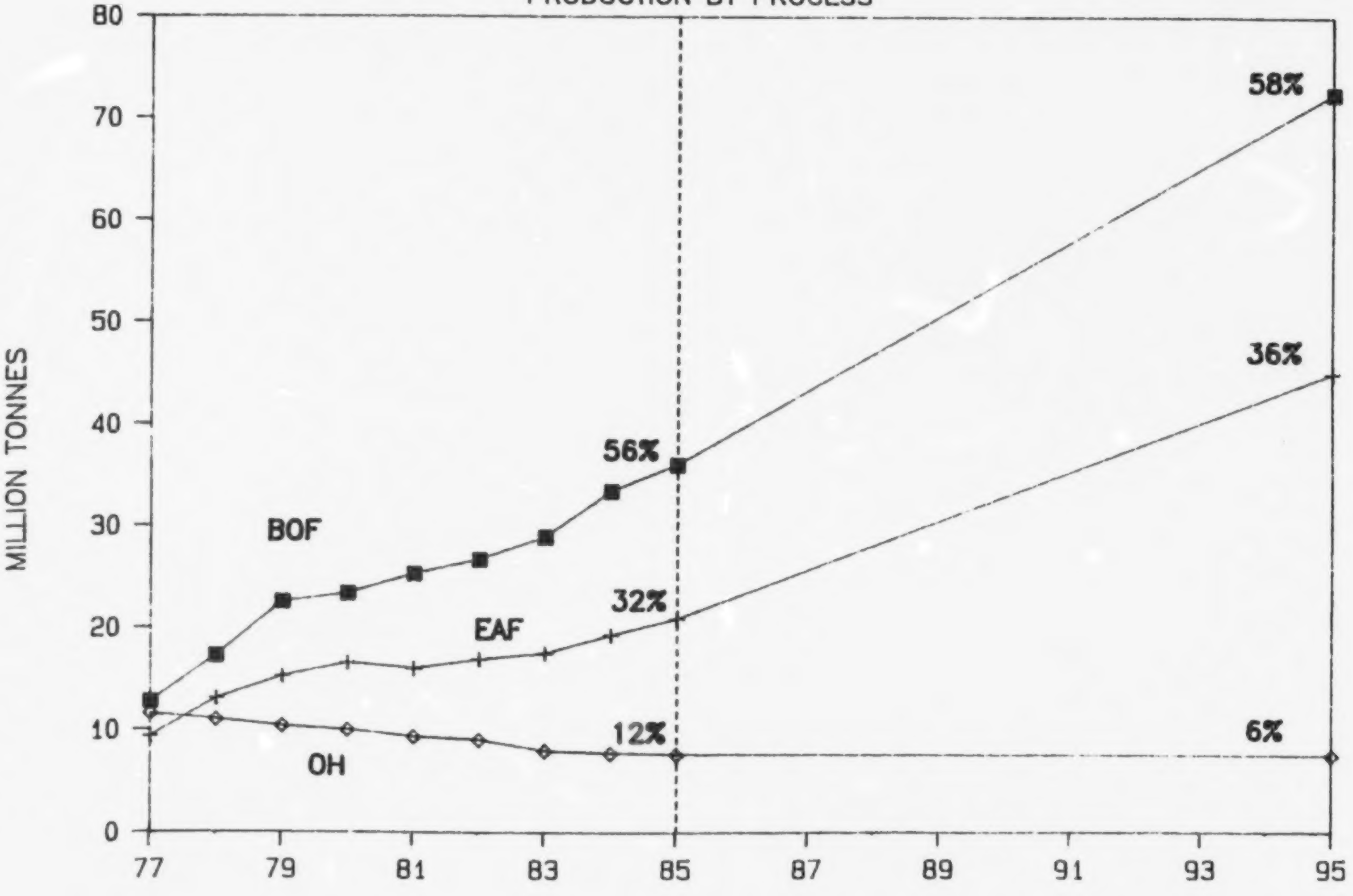
PRODUCTION BY PROCESS



GRAPH 5

MAJOR DEVELOPING COUNTRIES

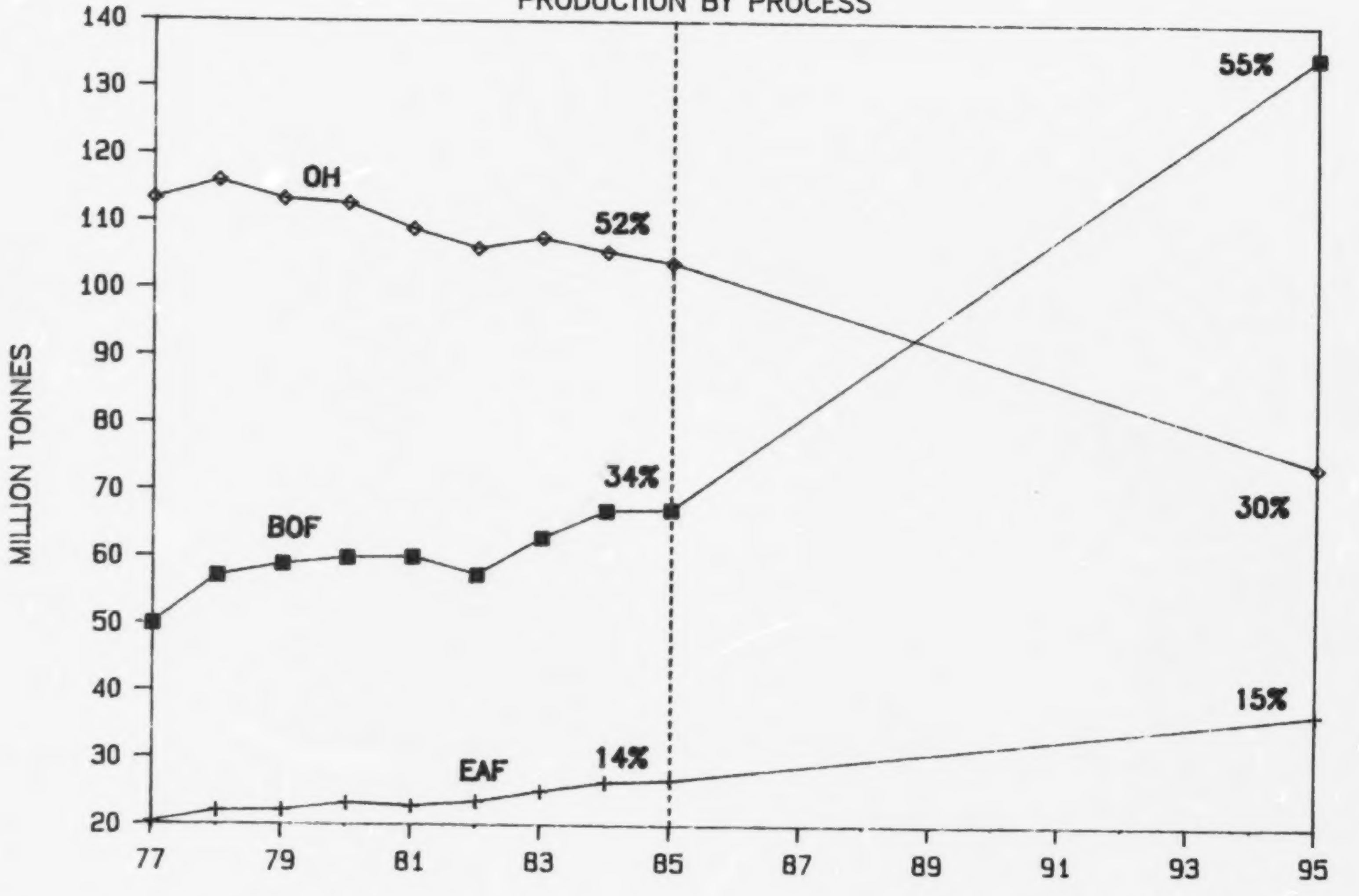
PRODUCTION BY PROCESS



GRAPH 6

E. EUROPE & USSR

PRODUCTION BY PROCESS



GRAPH 7

TABLE 4
Percent Steelmaking by Process - 1985

	OH	BOF	EAF
U.S.A.	7.3	58.8	33.9
Japan	-	71.0	29.0
Brazil	3.7	71.8	24.5
Australia	-	96.5	3.5
U.K.	-	71.1	28.9
Canada	-	72	28
Italy	-	47.5	52.5
Spain	-	38.9	61.1

The open hearth furnace share will decline and will not be a factor within the next 10 years.

Adoption of continuous casting has been one of the major recent steelmaking technology improvements. It has resulted in improved yields and reduced costs. Japan has adopted continuous casting faster than any other country. Currently it represents over 90% of Japan's crude steel production. In 1985 the percent continuously cast in EEC countries was 71% compared to 60% two years earlier. In 1985 continuous casting represented approximately 44% of crude steel output in the USA but it is expected to increase to about 80% by 1990. In 1986 it represented about 55% of output.

Developing Countries

In the early and mid seventies production of steel in the developing countries was split approximately one third for each steelmaking process - BOF, open hearth and electric arc furnace.

By 1985, the BOF and electric arc furnace share increased to 56% and 32%, respectively. The open hearth became much less of a factor at 12%. A further slight increase in BOF and electric arc furnace percentage is expected by 1995 at the expense of the open hearth process which will account for about 6%.

There will be a marked difference between countries as to the type of steelmaking process that will be selected for capacity additions. In those countries where a significant tonnage of flat rolled steel is required the BF/BOF will be the preferred route in spite of the high capital cost. However, when smaller increments of steelmaking capacity are needed the electric arc furnace will be chosen, particularly when non-flat products are to be produced. This scenario assumes that the power system is adequate to service the fluctuating heavy demand of the electric arc furnace. For flat products, virgin iron as DRI or hot metal, will be required as part of the charge to minimize the effects of impurities in the steel scrap. (Appendix 5 gives the view held by Midrex for the allowable levels of impurities - these are somewhat tighter limits than is thought necessary by others but do indicate the impurities of concern).

CPE Countries

The open hearth furnace has been dominant in the USSR and Eastern European countries, but will decline in share of production in favour of the more economic BF/BOF process. EAF share of production is quite low and is forecast to grow but not at the same rate as the BF/BOF.

China

Currently a significant percentage of steel production comes from major integrated steel complexes, including Anshan, Shanghai, Baoshan and Shoudou Works. Forecasts of up to 160 million tonnes by the year 2000 have been made based on construction of greenfield plants. However this forecast is now considered to be overly optimistic and the actual capacity may be closer to half this amount.

5.0 STEELMAKING CAPACITY

After the second world war the United States was the dominant steel producer. This position of leadership however was challenged by the EEC countries and the Japanese, primarily in the sixties, when significantly added capacity was installed. When worldwide consumption started to decline it led to retrenchment by the industrialized countries. At the same time developing countries were encouraged, in many cases with government blessing, to add capacity to achieve industrialization targets. In many cases the decision was influenced by a strategy of achieving export markets (primarily at the low end of the value added spectrum) which would offset the limitations of a small domestic market and provide needed foreign exchange.

The retrenchment of the industrialized countries is forecast to continue so that capacity will be more in line with consumption. The developing countries however are expected to continue adding capacity. Free World steelmaking capacity projections are shown in Table 5.

TABLE 5

Western World Steelmaking Capacity (1)
(million tonnes)

	1980	1985	1990	1990 (2)
W. Europe	223	195	170/180	185
N. America	152	139	110/115	120
Japan	142	125	110/115	120
Dev. Countries	60	86	110	110
Others	18	16	17	17
	595	561	517/537	552

Sources:

1. IISI, 1987
2. IISI, 1986

An estimate of 160.9 million tonnes of gross steelmaking capacity by the year 1995 for the developing countries has been made by Paine Webber Inc. in the June 1987 issue of World Steel Dynamics. Capacity by region is shown in Table 6.

TABLE 6

Developing World - Steelmaking Capacity

	1990		1995	
	%	(Million Tonnes)	%	(Million Tonnes)
Far East	37	(53.7)	43	(69.2)
Latin America	41	(59.5)	38	(61.1)
Africa	14	(20.3)	12	(19.3)
Middle East	8	(11.6)	7	(11.3)
	100	(145.1)	100	(160.9)

A detailed analysis of Far East countries with respect to present and anticipated steelmaking capacity, in tonnes, and expected shortfall in domestic consumption is shown in Table 7.

TABLE 7

	1984		1990		1995	
Country	Capacity	Capacity Shortfall	Capacity	Capacity Shortfall	Capacity	Capacity Shortfall
Hong Kong	0.25	- 1.89	0.25	- 2.55	0.25	- 2.95
Thailand	0.46	- 2.19	0.46	- 2.74	1.5	- 2.0
Singapore	0.75	- 1.55	0.75	- 2.15	0.75	2.65
Philippines	0.69	- 1.06	0.7	- 0.8	1.2	- 0.6
Malaysia	1.3	- 0.83	1.3	- 1.4	1.8	- 1.2
Bangladesh	0.25	0.02	0.25	- 0.05	0.25	- 0.15
Taiwan	3.25	2.7	5.7	- 2.3	10.0	- 1.8
India	15.0	1.95	17.5	- 4.0	18.0	-12.1
Korea	14.5	3.67	27.0	2.7	18.0	- 3.3

6.0 IRON UNITS

This section discusses, the two major iron units sources for the industry: iron ore and steel scrap. Both are major factors affecting steelmaking costs and hence affect melting process selection. The cost of these materials, hot metal, and selling prices of finished steel products is shown in Appendix 7.

Iron Ore

World production of iron ore is almost one billion metric tonnes. There is an abundance of iron ore around the world to meet the needs of the steel industry. The USSR is the largest producer followed by Brazil, Australia and China. Canada and the U.S.A. are also major sources of iron ore.

Iron ore is consumed primarily for the production of hot metal, DRI and, to a much smaller extent, as a direct addition to steel furnaces to control the temperature of the bath. The major iron ore consuming areas are Eastern Europe, Western Europe, Japan, North America and Latin America. The requirements of Africa and the Middle East are modest.

Western European countries, Japan and other Asian countries rely on imported ores hence transportation is a very important aspect of the supply equation. Ocean freight, together with the other shipping and handling charges, constitutes a significant part of the total cost of the ore to the consumer. It ranges from 25% to 50% depending on the source and market conditions.

Much of North America's iron ore industry is captive. Most of the USA integrated steel mills are so located that they cannot be served by large ocean vessels with lower cost iron ore from other sources.

To achieve added hot metal output and decreased operating costs, major technological improvements have been made to the blast furnace in the last decade. Burden preparation has been one of the most important developments. In North America the trend has been to use pellets (because of availability) whereas the European Community has favoured a feed consisting of sinter (primarily because of price) and a small quantity of pellets. The blast furnaces in Taiwan, Brazil and China operate almost without pellets, whereas the Japanese prefer about 10% pellets or high grade lump ore.

The operating cost of the major iron ore producers depends upon a number of factors such as size of the ore body, grade of ore, mineability, power costs, etc. These are beyond the scope of this report. Typical current prices of iron ore destined for the European market are shown in Table 8.

TABLE 8
FOB 1986 Prices European Market*
Dollars U.S.

Sinter Fines	
Carol Lake	.2650
Mt. Newman	.3240
CVRD (Brazil)	.2626
LKAB (Sweden)	.2860
Pellet Fines	
MBR (Brazil)	.2125
Pellets	
Carol Lake (Canada)	.3650
CVRD (Brazil)	.3560
LKAB (Sweden)	.3815

*Price per DMT (dry tonne) per 1% Fe

Source - The TEX Report

A price history of iron ore pellets at lake ports and from Brazil is shown in Table 9.

TABLE 9

Iron Ore Price

Year	Lake Superior Pellets Delivered at Rail of Vessel - Lower Lake Port US¢ Per Dry tonne Iron Unit	CVRD (Brazil) FOB US¢ Per Dry Tonne Iron Unit
1977	54.63	42.8
1978	59.95	37.5
1979	66.74	39.6
1980	72.55	47.1
1981	79.24	43.1
1982	78.95 - 85.54*	47.5
1983	79.24 - 85.54*	39.0
1984	79.24 - 85.54*	36.0
1985	57.1 - 85.54*	36.0
1986	57.1 - 85.54*	35.6
1987	57.1 - 85.54*	36.7

*Note- US¢85.54 prices denote Cleveland-Cliffs and M.A. Hanna Co. prices

The price of iron ore is primarily demand driven. Although the major suppliers would like to increase prices to cover their increased operating costs, it is going to be difficult in view of the major steel companies in Japan and Europe reducing their iron ore requirements and the current world over-supply situation. Consequently very little change is forecast in the price of this commodity.

Scrap Steel

Scrap steel is the primary charge material to the electric arc furnace. It is also added to the basic oxygen furnace as a percentage of the charge (20-30%) and likewise as part of the charge for the open hearth furnace (40-50%). It is part of the burden to the blast furnace (normally in the 1% range although in the U.S.A. it can exceed 4%).

In addition to the primary steel sector the foundry industry is a significant consumer of scrap. During the last decade the ratio of scrap to pig iron in ferrous foundries has been increasing on a world basis. In the U.S.A., it increased from about 65% in 1956 to almost 90% currently.

For the purpose of definition, there are three categories of scrap:

Home or Revert Scrap

Home scrap, or revert scrap, is that which is generated by steel mills and foundries in the production of new iron and steel (has been about 24% of raw steel production). Generally this category of scrap does not leave the mill or foundry but is reverted back to the furnace on site. In the U.S., home scrap has accounted for approximately one half of total scrap consumed annually, but is decreasing as a percentage of the total.

Industrial or Prompt Scrap

Industrial, or prompt scrap, is that which is generated by the metalworking industries with the automotive industry being the largest single source of this category of scrap, (33%). It accounts for 15% of finished steel consumed in industrialized countries. Prompt scrap is of known composition and chemistry and is often preferred by scrap consumers. It generally accounts for about 20% of the total scrap used annually in the U.S.A.

Obsolete Scrap

The remaining scrap used is composed of worn-out, unused, unwanted metal products - old automobiles, railroad cars, farm equipment, appliances, structural steel and the like. Obsolete scrap makes up the deficit between consumer demand and the available supply of industrial prompt scrap.

The 1985 consumption of scrap as compiled by IISI (excluding Latin America, Africa and Middle East, and Far East countries, other than Japan), was 256.3 million tonnes (Appendix 6). The countries excluded would likely account for another 73 million tonnes to give a world total of approximately 325 million tonnes. The major importing countries were Japan, Italy and Spain. The United States, U.K., France and Germany were the major exporting countries.

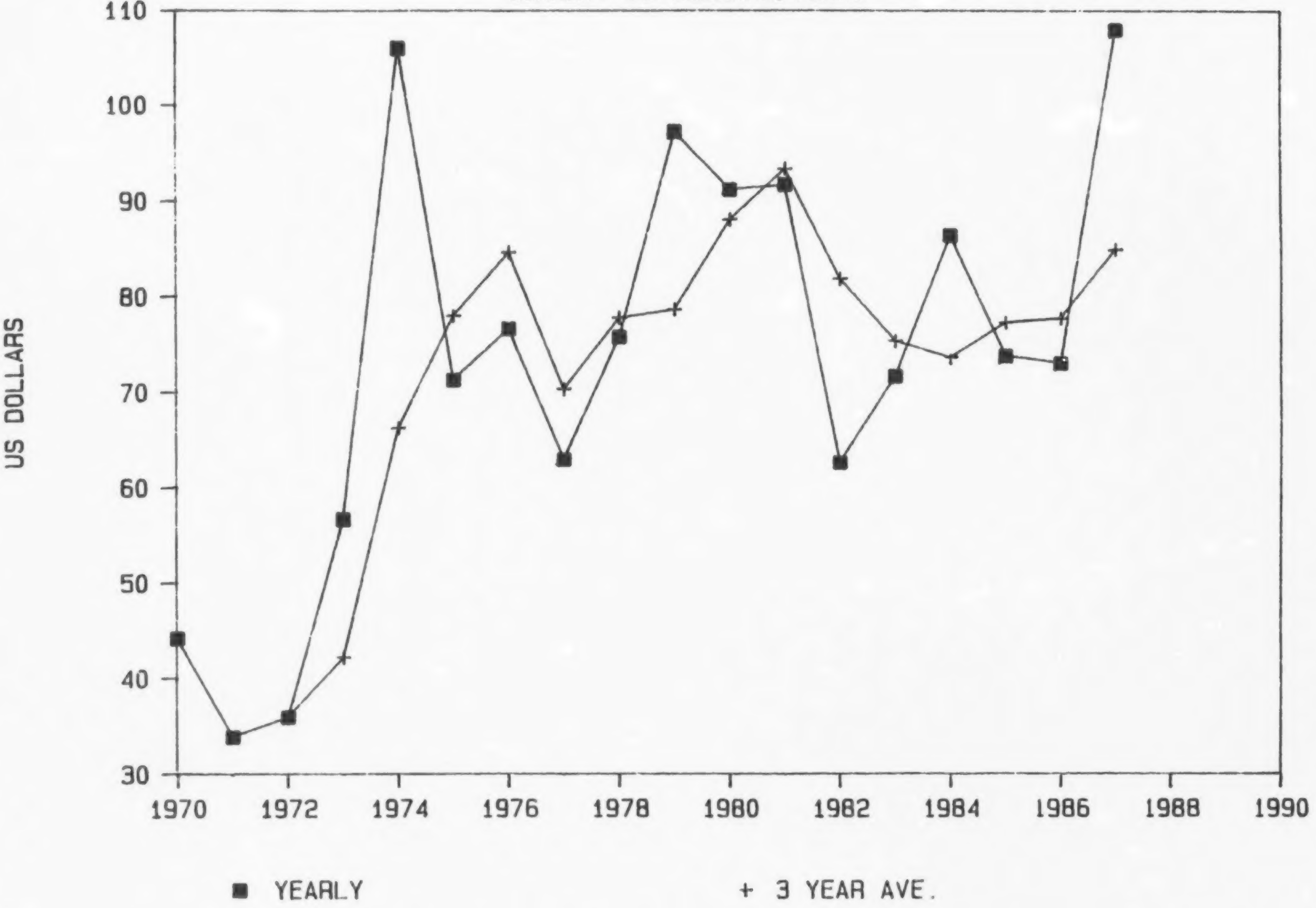
The yearly average price of composite No. 1 heavy melting scrap from 1970 to the present is shown in Graph 8 and in constant 1986 dollars in Graph 9.

The supply of home or revert scrap will decrease as a result of improved yields resulting from increased adoption of the continuous casting process. In addition, less industrial scrap will be generated because of improved manufacturing efficiencies of steel end products and the trend to less steel intensive equipment. The shortfall in supply will be made up by obsolete scrap which will have to bear the cost of inflation of labour and energy.

On the demand side of the equation, an increase in electric furnace production is forecast worldwide, and in particular in the U.S.A. Scrap steel will continue to represent a percentage of the BOF furnace charge. Less scrap, however, will be required by the open hearth operations as this process is being phased out.

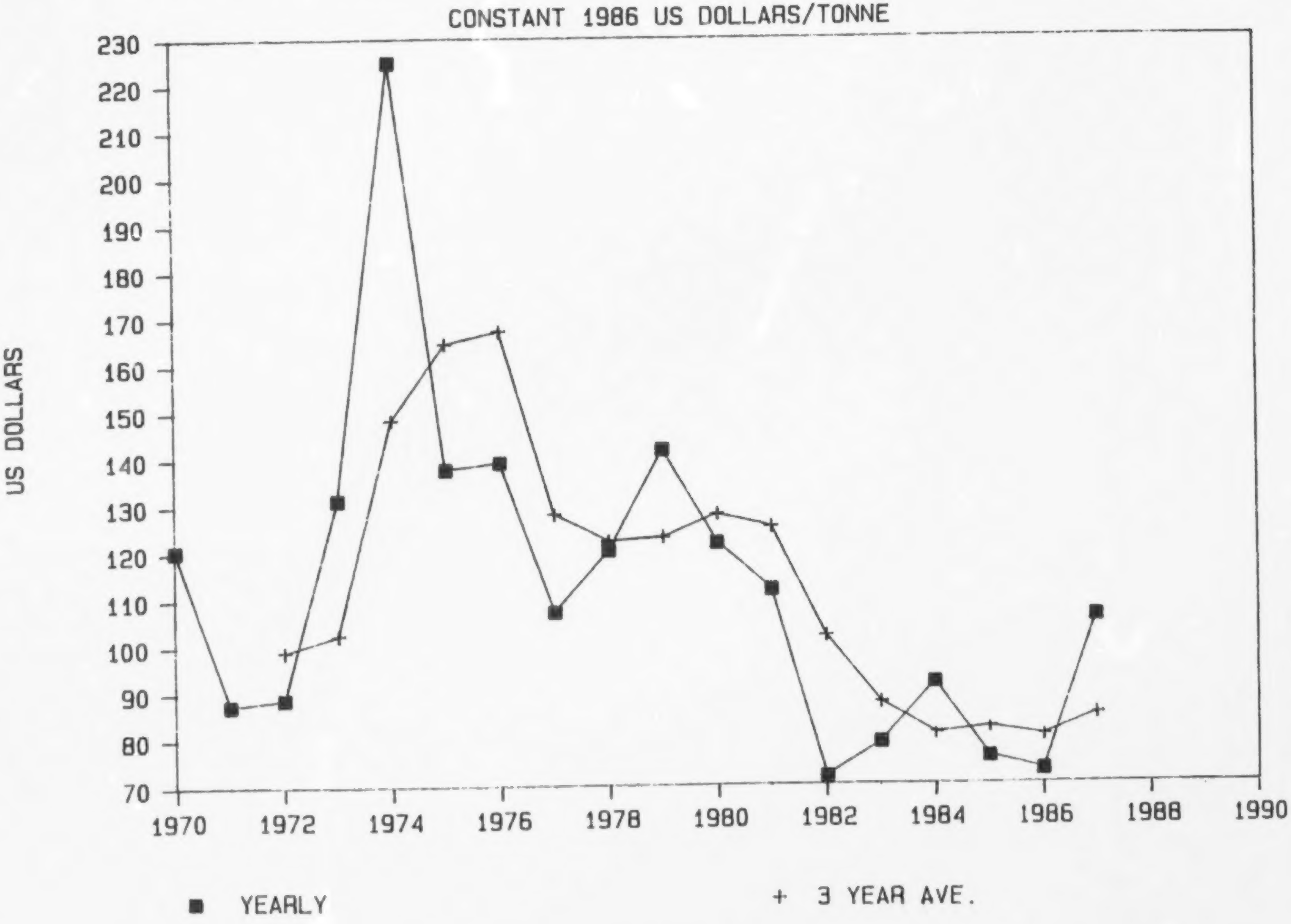
NO.1 HEAVY MELTING SCRAP PRICES

CURRENT US DOLLARS/TONNE



GRAPH [illegible]

NO.1 HEAVY MELTING SCRAP PRICES



GRAPH 9

Based on in-house information, the average composite price of No. 1 heavy scrap is forecast to increase and is expected to be in the $120 per long ton range by 1989, assuming the export market will require additional tonnages over 1987. Short term, monthly and quarterly fluctuations will continue as a result of short term demand pressures as has been the case in the past.

7.0 CONCLUSIONS

7.1 Overall Views

1. The current and the forecast market for steel, as determined by this study, supports some need for a process similar to that of the LB Process.

2. Despite pressures arising from environmental concerns about coking ovens and the overall large capital costs per annual tonne for steel produced via the BF/BOF route, it is fully expected that this will remain and perhaps increase its position as the dominant production process.

3. EAF production will continue to expand where electric energy is available and moderately priced.

4. In the future there will be more DRI despite the fact that the direct reduction process has not grown as quickly as expected. Increased growth can be expected if suitable scrap becomes less available, or if mini-mills develop the technology of producing flat rolled steel competitively, which will require a premium charge to the electric furnace.

5. The older open hearth process will continue to be replaced by either the blast BF/BOF or the EAF processes.

6. Steel has become more of a worldwide metal commodity with a current 30% of total production being traded internationally - up from a 17% level twenty years ago.

7.2 The LB Process and Specific Markets

The LB Process which is based on the potential economics of using finely sized iron ore, finely sized coal and the direct melting of sponge iron (by electric power) will be of interest where:

a) There is no ready source of coking coal but there is available low-cost coal suitable for direct reduction processes;
b) There is no low cost natural gas supply to produce DRI;
c) There is available low priced electrical power;
d) There is a need for relatively small increments of new capacity;
e) There is a shortage of low cost scrap steel;
f) There is a need to reduce the impurity levels of EAF steel (i.e., to dilute the transient impurities which enter the melt from scrap).

In the industrial countries a possibility does exist for increased DRI and therefore, potentially, for LB produced iron. If the mini-mill installations based on EAF technology are coupled with lower cost rolling facilities, made possible by improved thin-slab casting processes, then there will be a need for higher quality input material to meet the chemical specifications of flat-rolled products. DRI or equivalent material (via LB or other processes) could, under these circumstances, fulfill this increased market need wherever the new mini-mill installations (with smaller capital costs) can challenge the large integrated steel companies flat-rolled markets.

In the developing countries, in China and to some extent in the CPE countries, the same arguments hold as for the industrialized countries. However there is expected to be more change in the steelmaking facilities in these regions (more overall production and/or a shift away from open hearth production) and hence perhaps a greater opportunity for DRI and the LB Process. The potential market for the LB Process will be influenced as follows:

- If steel requirements are being satisfied by modern blast furnaces and the BOF process, the possibility of introducing a new process is remote unless there is an economic incentive;
- Electric arc furnaces will be the preferred melting method for the low value added products when the production required is considerably less than that of a BF/BOF operation. Iron units will be obtained from scrap steel;
- For flat products, in particular deep drawing grades, a percentage of the charge must be virgin iron units. These virgin iron units can be supplied by commercial installations such as Midrex and HYL where natural gas is attractively priced because of availability;
- For those countries where natural gas is not available or required for other end uses, the LB Process would merit consideration particularly if the country has suitable coal deposits.

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APPENDIX 1

COUNTRIES BY REGION

Industrialized Nations = W. Europe + Canada
U.S.A., Japan, South Africa, Australia

CPE = U.S.S.R. + Eastern Europe

Developing = All other countries

World = Ind. + CPE + Developing + China

APPENDIX 2

Import and Export Statistics

APPENDIX 2A

WORLD STEEL TRADE SIGNIFICANTLY INCREASES AS A PROPORTION OF THE WORLD STEEL PRODUCTION

(Million Tonnes of Crude Steel and Crude Steel Equivalent)

Year	Total Exports*	Total Production	Export (%)
1950	20.5	192.0	10.7%
1955	34.0	270.5	12.6%
1960	52.7	345.5	15.3%
1965	78.5	457.0	17.2%
1970	117.5	595.4	19.7%
1975	148.1	643.3	23.0%
1980	179.2	717.1	25.0%
1985**	216.5	719.0	30.1%

Source: IISI

* For consistency the recorded export/import trade of steel has been converted to an equivalent tonnage of crude steel (required for the production of the semi-finished and finished product which are traded) using a factor of 1.3.

** In the past decade the rapid increase in crude steel being available as continuously cast steel has actually reduced the equivalent conversion factor to approximately 1.25 as about 50% of the world's steel is now made by this more efficient process. If this conversion factor was used for 1985 the percentage of trade to total production would be closer to 29% rather than 30%.

APPENDIX 2B

WORLD STEEL TRADE BY AREA
Exports, Imports and Net Trade Balances
(Million Tonnes Product Tons)

Industrialized Countries	1970	1975	1980	1985
+ Exports	72.9	94	111.7	126.1
- Imports	58.0	57.3	71.8	83.0
Net Trade	+ 14.9	+ 36.7	+ 39.9	+ 43.1
CPE				
+ Exports	15.0	17.3	18.0	20.9
- Imports	11.5	20.7	19.2	20.9
Net Trade	+ 3.5	- 3.4	- 1.2	0
China				
+ Exports	0.2	0.4	0.4	0.2
- Imports	2.7	4.0	5.0	19.2
Net Trade	- 2.5	- 3.6	- 4.6	- 19.0
Developing Countries				
+ Exports	2.3	2.3	7.7	19.3
- Imports	18.0	32.0	41.8	43.4
Net Trade	- 15.7	- 29.7	- 34.1	- 24.1
Total Exports	90.2	114.0	137.8	166.5
Total Imports	90.4	114.0	137.8	166.5
If 1970 = 100%	100%	127%	153%	185%
Net Exporters	18.7	36.7	39.9	43.1
Net Importers	18.5	36.7	39.9	43.1
If 1970 = 100%	100%	202%	219%	237%

APPENDIX 3

Steel Consumption by Product - Thailand, 1973 - 1982

APPENDIX 3

STEEL CONSUMPTION BY PRODUCT - THAILAND

1973 - 1982
(x 1000 tonnes)

	1973	1974	1975	1976	1977	1978	1979	1980	1981	1982
Railway/Rail	-	-	-	-	-	5	23	1	1	1
Section	85	82	72	87	124	161	127	93	136	107
Bar	540	447	411	501	584	633	681	610	567	470
Wire Rod	-	76	76	100	123	133	141	129	128	122
Wire Product	28	28	25	29	51	22	22	17	14	14
Subtotal	653	633	584	717	882	954	994	850	846	714
Sheet/Coil	147	123	64	251	232	336	414	363	514	529
H&M Plate	56	55	82	80	92	35	82	58	40	37
Tin Plate	70	58	46	71	96	88	131	97	141	114
Galvanized Sheet	102	94	112	130	160	139	222	146	171	170
Other Coated Sheet	19	22	12	10	23	9	14	8	10	16
Subtotal	394	352	316	542	593	607	863	672	876	866
Seamless Tubes	11	8	3	3	5	2	7	11	24	9
Welded Pipes	118	116	126	170	160	138	147	144	124	116
Subtotal	129	124	129	173	165	140	154	155	148	125
TOTAL	1176	1109	1029	1432	1640	1701	2011	1677	1870	1705
% Flat Rolled Products	34	32	31	38	36	36	43	40	46	51

APPENDIX 4

Listing Electric Furnaces by Country

WORLD ELECTRIC ARC FURNACE REFERENCE LIST

This list gives details of electric furnaces understood to be currently in operation in steelworks. Foundry arc furnaces are in most cases not listed. Where a works is known to have electric arc furnaces but details are not available, the works' name only is shown. Capacities are given in tonnes except where stated otherwise.

	No.	Size (tonnes)	Capacity (tpy)
ALBANIA			
Enver Hoxha Tractor Plant, Tirana	1		
Steel of the Party Metallurgical Combine, Elbasan	3		
ALGERIA			
SNS, El Hadjar	1	85	
ANGOLA			
Siderurgia Nacional, Luanda	1	18	30,000
ARGENTINA			
Acindar SA, Villa Constitución	3	110	921,000/ 1,350,000
	2	63	200,000
La Tablada, San Justo	2	25	260,000
	1	20	
	1	50	
Avellaneda	2	65	240,000
Aceros Bragado, Bragado			
Cura Hnos, Rosario			
Dalmine Siderca, Campana	2	50	400,000
	1	65	
	1	75	
Zapla, Palpala	2		70,000
AUSTRALIA			
BHP,			
Port Kembla	1	50	100,000
Whyalla	1	19	30,000
Comsteel,			
Waratah	1	50	
	1	30	
	1	20	
	1	10	
Moorooka	1	3	
SCI,			
Laverton North	1	4.5 metres dia	200,000
AUSTRIA			
Breitenfeld, Mitterdorf	1	8	80,000
	1	15	
	1	30	
Marienhütte, Graz	1	25	100,000
VEW,			
Kapfenberg	4	15/45	
Ternitz	3	20/50	
Voest-Alpine			
Linz	2	25	74,000
Liezen	2	4/7	
Traisen	1	4	20,000
	1	7	
BELGIUM			
ALZ, Genk	1	35	120,000
Boël, La Louvière	1	8	
Fabrique de Fer de Charleroi, Marchienne			
Henricot, Court-Saint-Etienne	3	10	
Jemeppes, Jemeppes			
Tubemeuse, Seraing	1	5.8 metres dia	400,000
BRAZIL			
Acesita, Timóteo	1	8	150,000
	1	28	
	1	30	
Açonorte, Recife			
Açopalma, Vàrzea da Palma	1	10	10,000
Aliperti, São Paulo	1	25	60,000
Anhanguera, Mogi das Cruzes	2	40	
	1	70	
Aparecida, Sorocaba	1	6.5	167,000
	1	10.8	
	1	13	
	1	21.4	
Cearense, Maranguape	1	10	
Cobrasma, Osasco	1	15	55,100
	2	6	38,400
	1	12	30,600
Cofavi, Cariacica	1	3.4 metres dia	51,000
	2	3.6 metres dia	62,000
Comesa, Atalaia	1		
Cosigua, Santa Cruz	2		545,000
Cosim, São Paulo	1	8	
	1	1	
Cosinor, Recife-Pernambuco	1	10	
	1	5	
	1	4	
Dedini, Piracicaba	1	30	152,000
	1	20	
	1	8	
	1	6.5	
	1	1.3	
Eletrometal, Sumaré	1	22	40,000
	2	11	60,000
Fi-El, S. José dos Campos	2	15/17	120,000
Guaira, Curitiba Araucária, PR	1	80	240,000
Hime, Rio de Janeiro	1	55	200,000
Itaunense, Itaúna	1	10/12	57,500
	1	12/14	62,500
Mannesmann, Barreiro	3	40	230,000
Mendes Júnior Juiz de Forat			
Aços Finos Piratini, Charqueadas	1	40	
	1	12	
Riograndense SA, Porto Alegre			
Santa Olimpia, Iparanga and São Paulo	3		96,000
Sidelpa, Lençois Paulista	1		40,000
Usiba, Simões Filho	1	120	240,000
Villares	3		270,000
Aços Villares, São Caetano do Sul	2	25	100,000
	1	15	30,000
	1	5	20,000
	1	4	8,000
	1	2.5	9,000
BULGARIA			
Kremikovtsi	1	100	135,000
	2	100	
Lenin Iron & Steel, Pernik	4	10	550,000
	2	100	
	2	100 (1985)	
CANADA*			
Atlas, Welland, Ont	2	60	325,000
	1	25	
Tracy, Que	1	60	70,000
Burlington, Hamilton, Ont	2	40	300,000
	1	70	
Canadian Steel Wheel, Montreal, Que	2	60	160,000
Courtice, Cambridge, Ont	1	25	85,000
Crucan Div, Slater Steel Industries	1	40	30,000
	1	15	10,000
Ipsco, Regina, Sask	2	120	700,000
	1	90	250,000
	1	40	50,000
Ivaco, L'Orignal, Ont	2	65	
Lasco, Whitby, Ont	2	65	455,000
	1	130	455,000
Manitoba, Selkirk, Man	2	50	280,000
Sidbec-Dosco, Contrecoeur, Que	2	100	550,000
	2	150	800,000
Montreal, Que	1	50	175,000
Longueuil, Que	2	50	300,000
Stelco, Contrecoeur, Que	1	90	
Edmonton, Alta	2	75	
Western Canada, Vancouver, BC	1	80	150,000
Calgary, Alta	1	40	120,000
CHILE			
Indac, Santiago	1	12	
Cia de Acero, Rengo	1	10	
CHINA			
Anshan Works	8		
Daye Works	1	5	
	7	20	
Echeng Works	3	5	
Fushun Works	1	60	
Hefei Works	1		
Shanghai Works	4	10	
	5	25	
	2	10	
Shoudou Works	3		
Taiyuan Works	5	5	
	2	50	
COLOMBIA			
Metalúrgica Boyacá SA, Tuta Paz del Río, Boyacá	1	20	
Sidelpa, Cali	2	14	70,000
Sidemuña, Bogotá	1	7	
	1	10	
Simesa, Medellín	1	12ft 6in dia	85,000
	1	9ft 10in dia	15,000
CZECHOSLOVAKIA			
NHKG, Ostrava	1	60	105,000
Poldi, Kladno	2	100	
Vitkovice, Ostrava	1	75	

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APPENDICES

1. Countries by Region

2. Import and Export Statistics

3. Steel Consumption by Product - Thailand, 1973 - 1982

4. Listing Electric Furnaces by Country

5. Maximum Impurity Levels

6. Consumption of Scrap, 1985 - IISI

7. Comparative Prices - Iron Ore and Steel Products

8. Steel Statistics - Developing Countries

APPENDIX 1

Countries By Region

ELECTRIC ARC FURNACES

	No.	Size (tonnes)	Capacity (tpy)
DENMARK			
Det Danske Stålvalsevaerk, Frederiksvaerk	2	100	
DOMINICAN REPUBLIC			
Metaldom	1	20	40,000
	1	16	25,000
	1	20	35,000
DUBAI			
Ahli Steel, Al-Ramool			
ECUADOR			
Ecuasider, Machala	1§		
EGYPT			
Alexandria National Steel Co, El Dikheila	4§	70	
Delta Steel Mill, Mostorod	5		
Egyptian Copper Works	1	5 }	45,000
	1	25 }	
Egyptian Iron & Steel	2	12	
National Metal Industries, Kalyoubia	2	35	150,000
EL SALVADOR			
Corinca, SA, Quezaltepeque	1	6	10,000
Siderurgica Centroamericana del Pacífico, Sonsonate	1	50	100,000
Siderúrgica Salvadoreña Tinetti, Ciudad Delgado	1	4	4,000
FINLAND			
Outokumpu, Tornio	1	60	150,000
Ovako, Imatra	1	60	
	1	40	
Dalsbruk	1	35	
FRANCE			
Allevard, Le Cheylas	1	30	120,000
Alpa, Porcheville	1	60	180,000
Aubert & Duval, Les Ancizes	4	3/60	
Bonpertuis, Rives sur Fure	2	3	
Champagnole, Champagnole	1	10	12,000
Cockerill-DRC, Hautmont	1	45	180,000
Commentryenne, Commentry	1	30	15,000
	1	18	10,000
Creusot-Loire, Le Creusot	2	100	
Rive-de-Gier	1	80	
	1	90	
Ondaine Works, Firminy and Unieux	1	30	
	1	40	
Pamiers	1	40	
Fives-Cail Babcock, Denain	1	50 }	100,000
	1	35 }	
	1	10 }	
CFAS, Les Dunes	1	45	80,000
	1	75	300,000
Le Marais, St. Etienne	1	80	150,000
Imphy SA, Imphy			
Iton Seine, Bonnières sur Seine	1	45	250,000
SAM, Montereau	1	85	400,000
Pompey, Pompey	1	80	
	1	10	
Ugine Aciers, Ugine	2	40	160,000
Fos	1	120	300,000
L'Ardoise	1	40 }	180,000
	1	60 }	
	1	45	
Usinor, Thionville	1	70	200,000
Isbergues	2	95	400,000
Sedan	1	18	
Thionville	1	70	200,000
Vallourec SA, Anzin, Saint-Saulve	1	70	
EAST GERMANY			
Brandenburg	2	120	550,000
Freital			240,000
Wilhelm Florin, Hennigsdorf	1		175,000
	2		
Riesa, Riesa/Elbe			
WEST GERMANY			
Arbed Saarstahl, Völklingen	1	100	396,000
	1	30	48,000
	1	20	32,000
Badische, Kehl	2	60	826,000
Benteler-Werke, Paderborn			
Berliner Stahlwerk, Berlin	1	45	120,000
Bochum, Bochum	1		74,000
Böhler, Düsseldorf	1	20	
Boschgotthardshütte, Siegen	1	42	
	1	22	
Bous/Saar, Bous	1	60	200,000
Buderus, Wetzlar	1	40	
EBV, Eschweiler	2	40	180,000
Hamburger Stahlwerke, Hamburg	1	110	660,000
	1	90	
Hoesch Hüttenwerke Dortmund	1		190,000
Klöckner-Werke Osnabrück	2	100	348,000
	1	20	24,000
	1	10	12,000
Krupp Stahl, Bochum Siegen	1	120	600,000
	3		800,000
Lech, Meitingen-Herbertshofen	3		450,000/500,000
Lemmerz-Werke, Königswinter	2		
Mannesmannröhren-Werke, Duisburg-Huckingen	1	50	
Moselstahlwerk, Trier	1	25/30	
	1	35	
Schmidt & Clemens, Lindlar	1	8	10,000
Thyssen Edelstahlwerke, Krefeld	2	75	
	1	30	
	1	15	
	1	4	
Witten	1	110	
Thyssen Henrichshütte, Hattingen	1	150	
	1	40	
Thyssen Niederrhein, Oberhausen	2	120	
GREECE			
Halyps, Volos	2	50	300,000
Halyvourgiki, Eleusis	3	100	850,000
Helliniki Halyvourgia, Aspropyrgos	2		200,000
Sidenor, Salonika	1	50 }	450,000
	1	30 }	
Thessalias, Volos	2	45	350,000
GUATEMALA			
Hornos, Guatemala City	2	8ft 9in	36,000
	1	9ft 10in	24,000
HONDURAS			
Aceronsa, San Pedro Sula	1		24,000
HONG KONG			
Shiu Wing, Junk Bay	1	45/50	
Shun Fung, Junk Bay	4		
HUNGARY			
Csepel, Csepel Island	3		
Danube Works	2	5	
Diosgyör, Diosgyör	1	80	
	1	50	
	2	15	
	2	10	
INDIA			
Andhra Steel Corp, Malkapuram, Visakhapatnam	1		25,000
Bangalore	1		50,000
Ashoka Alloy, Ambala City	1	5	7,000
	1	17	23,000
BD Steel Castings, Tarapur	1	10/12	20,000
BDA Steels	2	5	
	1	3	
Bhagwan Industries, Vaid Steels, Aishbagh	1	10/12 }	40,000
	1	5 }	
Bhoruka Steel, Bangalore	3		80,000
Bihar Alloy, Hazaribagh	2	32	
Canara Steel, Mangalore	1	10	18,000
Dabriwala Steel & Engineering, Faridabad	2	4/5	20,000
	1	10/12	36,000
Grand Alloys & Steels, Hooghly	1	12	27,000
Guest Keen Williams, Howrah	2	5 }	85,000
	1	14 }	
	1	15 }	
	1	20 }	
Hope (India), Pench Steels Div, Sanagunj	1	10/12	18,000
	1	10/12	18,000
Ellora Steels, Aurangabad	1	5	12,000
Ennore Steel, Madras	1	5	7,000
Facor, Nagpur	1	10	18,000
Indo-Japan Steels, Howrah	1	3	6,000
Indore Steel & Iron, Indore	3		48,500
Jindal Iron & Steel Co Ltd, Boisar	1	10	21,000
Kalyani Steels, Poona	2	10/12	
Kanoria Steels, Boisar	2	6/8	22,000
Kap Steel, Bangalore	1	15/17 }	48,000
	1	15/17 }	
	1	5 }	
Maharashtra, Ghaziabad	1	10/12	
Mahindra Ugine, Jagdish Nagar, Khopoli	1	20	
	1	40	
Man Industrial, Jaipur	1	4.5	
Modella Steels & Alloys, Ratlam	2	12	40,000
Modi Industries, Modinagar	3	5	108,000
	3	10	
Mohan Steels, Unnao	1	12	22,500
Mukand, Thane	4	30	135,000
Muzaffarnagar Steels, Muzaffarnagar	1	10/12	18,000
Panchmahal Steel, Panchmahals	2	10/12	36,000
Partap Rajasthan Special Steels, Jaipur	2	10/12	36,000
Partap Steel Rolling Mills, Chheharta	2	10/14	
	1	15/17	
Ballabgarh	3	5/7 }	84,000
	1	10/14 }	
	1	15/17 }	
Patancheru	1	10/14 }	55,000
	1	18/20 }	

	No.	Size (tonnes)	Capacity (tpy)
India Continued			
Partap Steels, Mandi Gobindgarh	1	5/7	10,000
Punjab Concast, Ludhiana	2	12	50,000
	1	5	
	1	16	
Punjab Iron & Steel, Jullundur Cantt	1	3	12,000
	1	5	
Raipur Wires & Steel, Raipur	3	10/14	54,000
Rathi Alloys & Steel, Dasula	1	25	100,000
	2	10/12	
Ghaziabad	1	12/14	
Rathi Ispat, Ghaziabad	2	10/12	40,000
Sail, Alloy Steels Plant, Durgapur	3	50	
	1	10	
Sanghvi Steels, Taloja	1	12	18,000
Saroj Alloys & Steels, Kariganur, Hospet	1	10	18,000
Singh Alloys & Steel, Kalyani	1	10	20,000
Boisar	2	10	40,000
Singh Electrosteel, Tarapur	2	15	50,000
Kalyani	2	15	50,000
Somani Steels, Unnao	2	10/12	36,000
Star Steel, Baroda	1	10/14	21,000
	1	10/14	21,000
Steel Complex, Calicut	3	10/12	50,000
Steel Ingots, Dewas	1	5/7.2	10,000
Tamil Nadu Steels, Arakkonam	1	25	36,000
Tata Iron & Steel, Jamshedpur	2	10	31,000
Trichy Steel Rolling Mills, Senthannipuram, Tiruchirapalli		9	16,000
		5	
		2	
Universal Steel & Alloys, Faridabad	1	8	56,000
	2	10/12	
Upper India Steel Mfg & Eng, Ludhiana	2	5	25,000
	1	5	
Usha Alloys & Steels, Adityapur, Jamshedpur	1	20/25	50,000
Visvesvaraya, Bhadravati	2	20	44,400
	1	8	15,600
	1	6	6,900
	1	3.5	4,000
INDONESIA			
Baja Indonesia Utama, Jakarta	3	25	150,000
Ispat Indo, Surabaya	1	80	200,000
Krakatau Steel, Cilegon	4	65	540,000
	4	130	1,060,000
Pulogadung, Jakarta			
IRAN			
National Iranian Steel Corp, Ahwaz	6	60	1,600,000
IRAQ			
Khor al Zubairs	4	70	
IRISH REPUBLIC			
Irish Steel, Haulbowline	1	90	345,000
	1	30	90,000
ISRAEL			
United Steel Mills, Haifa	2	60	300,000
ITALY			
Ads, Casoria			
Atem, Campofelice di Roccella			

	No.	Size (tonnes)	Capacity (tpy)
Beltrame, Vincenza	1	90	350,000
Alfa, Brescia			
Acciaierie Ferriere Alpine, Borgone di Susa	2		
Ferriera Alto Milanese, Caronno Pertusella			
Arvedi, Cremona	2	40	
ASO, Ospitaletto	1	30	50,000
Fratelli Bertoli, Udine			
Acciaierie di Bolzano, Bolzano	2	45	200,000
Ferriere di Borgaro SpA			
Busseni, Nave			
Acciaieria e Ferriera del Caleotto, Lecco	1	35	120,000
Calvisano, Calvisano			
Caronno, Caronno Pertusella			
Ferriera Acciaieria Casilina SpA, Pantano Borghese	1	16	30,000
Ceretti Pallanzeno			
Ferriera de Ceto Spa, Ceto	2		
F. di C. Fonderie di Chiari SpA, Chiari	3		
Acciaieria Fonderia Cividate SpA, Cividate del Fruili	3		
Acciaiere di Cividate al Piano SpA, Cividate al Piano	3		
IAI Cofermet Industria Acciai Inox, Turin	1		300,000
Cravetto, Bruzolo	1	40	100,000
	1	50	120,000
Crema P. Stramezzi, Crema	1	40	
Dalmine, Dalmine	1		156,000
	1		200,000
Darfo, Darfo			
Falck SpA, Sesto S. Giovanni	1	140	400,000
	2	60	300,000
	2	135	500,000
Fenotti & Comini, Nave			
Feralpi, Lonato	3	30	300,000
Ferio, Rivoli di Osoppo			
Ferreo, Turin			
FIT, Riva Trigoso	2	54	200,000
	1	35	
Foroni, Gorla Minore	1	35	
Galtarossa, Verona			
Ilfo, Odolo			
Indumetal, S. Giorgio di Nogaro	1	50	180,000
Iro, Odolo			
Nuova Italsider Genoa-Campi	1	115	350,000
	2	40	
Luigi Leali, Odolo	2	22	180,000
	1	25	100,000
Lonato, Lonato			
Lucchini, Settimo Torinese			
Siderurgica Meridionale Stefana Antonio, Termoli			
Acciaierie e Tubificio Meridionali, Bari			40,000
Meroni, Erba			
Modena SpA, Modena			
Ferriere Nord, Osoppo	1		350,000
Ola-Officine Laminatoi Sebino SpA, Pisogne	2	60	
Ori Martin, Brescia	3	30	200,000
Nave, Montirone			
Pietra SpA, Brescia Omegna	2		
Acciaiere di Porto Nogaro SrC	2		

	No.	Size (tonnes)	Capacity (tpy)
Predalva Acciaieria e Ferriera Srl, Piancamuno	2		
Ferriera Ernesto Preo & Figli SpA, Marghera	3		
Prolafer, Trino Vercellese			
Pugliesi, Giovinazzo	1	35	70,000
Raimondi SpA, Rescaldina	1	18	35,000
Redaelli Sidas-Siderurgica Acciai Speciali SpA, Milan	2		
Acciaieria di Rubiera SpA, San Donnino di Casalgrande	3		
Metallurgica Rumi SpA, Montello	2		
Safas-Stà Azionaria Fonderia Acciai Speciali SpA, Tavernelle di Attavilla	4		
Safau, Udine	2	80	
Acciaierie San Zeno SpA, San Zeno Naviglio	4		
Sant'Anna, Sesto Calende			
Siderurgica Commercial Santo Stefano, Pero			
Seta, San Zeno Naviglio	3	90	700,000
Nuova Sias, Aosta	1	80	
	2	50	
	1	10	
Milan	2	60	400,000
Sideradria SpA, Adria			
Sisma, Villadossola	1	40	70,000
	2	50	180,000
Siava SrL, Calvisano			
Michetti, Sovere			
Stefana Antonio, Brescia	1	28	80,000
Stefana Fratelli fu Girolamo, Nave and Ospitaletto			
Stramezzi, Crema	1	40	
Tanaro, Lesegno			
Tassara, Breno			
Terni, Terni	2	180	910,000
	2	45	
	1	85	
	1	100	
Lovere	1	40	145,000
	2	6	20,000
Valbruna, Vicenza	1	40	90,000
	1	10	25,000
Piove di Sacco	1	10	25,000
Valsabbia Odolo and Sabbio Chiese			
Valsugana, Borgo Valsugana			
Venete, Padua			
JAPAN			
Aichi, Chita	1	80	1,050,000
	2	50	
	1	30	
	2	10	
Azuma, Sendai	2	100	840,000
Daido, Chita	4	70	1,200,000
Hoshizaki	2	15	100,000
	1	30	100,000
Shibukawa	2	15	91,000
	1	25	96,000
Daiwa Denki, Seiko, Kobe			
Funabashi, Funabashi	1	50	250,000
	1	100	590,000
Godo Steel, Tokyo	1	60	220,000
Himeji	1	60	176,000
	1	70	345,000
Japan Steel Works, Muroran	1	10	
	1	25	
	1	100	
	1	120	
Kansai, Sakai	1	15	
	1	30	
Kawasaki, Mizushima	1	30	72,000
Chita	2	15	108,000

ELECTRIC ARC FURNACES

	No.	Size (tonnes)	Capacity (tpy)
Japan Continued			
Kobe, Kobe	1	60	88,000
	1	70	183,000
Takasago	1	60	
	1	25	
	2	15	
	1	100	
Kokko, Osaka	1	30	
Kurimoto, Kagaya	2		
Kyoei, Hirakata	1	90	
Nachi-Fujikoshi, Toyama	4	30/7/5	
Nakayama Steel Products, Nishiyodogawa-ku	1	70	360,000
Niigata Myodo Metal, Niigata	2	10	52,000
Nippon Koshuha, Toyama	1	40	186,000
	2	10	30,000
Hachinohe	4		
Nippon Metal Industry, Sagamihara	2	50	
	1	15	
Kinuura	1	70	
Nippon Stainless, Naoetsu	1	30	48,000
	1	·15	26,000
Nippon Steel, Hikari	1	40	289,000
	1	60	
Nippon, Yakin, Kawasaki	2	10	325,000
	1	30	
	2	60	
Kanazawa	1	3	3,000
Nisshin, Shunan	1	6.5	14,000
	4	30	362,000
NKK, Keihin	1	50	
Pacific Metals, Hachinohe	1	30	75,000
Rinko, Osaka	1	40	180,000
	1	15	60,000
Sanyo Special Steel, Himeji	1	70	
	2	30	
Seibu Chemical, Saitama	1	75	
Sumitomo Electric, Itami	2	20	130,000
	1	6	18,000
Sumitomo Metal Industries, Wakayama	1	80	168,000
Osaka	1	2	263,000
	1	8	
	1	15	
	1	25	
	1	80	
Amagasaki	1	8	102,000
	1	50	
Tokyo Steel Mfg, Okayama	2		1,440,000
Takamatsu	2		600,000
Kitakyushu	2		610,000
Kochi	2		240,000
Tokyo Tekko, Oyama	2	50	345,600
	1	70	518,400
Hachinohe	1	70	518,400
Topy Industries, Toyohashi	1	30	120,000
	2	30	288,000
	1	120	360,000
Toshin, Tokyo	1	70	300,000
	1	70	300,000
Himeji	1	50	280,000
	1	50	280,000
	1	150	700,000
Ube, Ube			
Yamato Kogyo, Himeji			
Yodogawa, Osaka	2	20	120,000
	1	8	
JORDAN			
Jordan Iron & Steel Industry Co. Ltd,			
KENYA			
Kenya United Steel, Mombasa	2	4/5	18,000
Steel Billet Castings, Nairobi			20,000
NORTH KOREA			
Hwanghai Iron Works, Songnim			
Kangson Works, Kangson	8		960,000
Kimchaek Works, Kimchaek			
SOUTH KOREA			
Daehan Sangsa, Pusan			120,000
Dongkuk, Pusan	3	15	216,000
	2	40	500,000
	1	50	280,000
Inchon	2	30	290,000
Inchon, Inchon	1	70	
	2	60	1 230,000
	2	45	
	1	30	
	1	10	
Kangwon Industrial, Seoul	2	30	200,000
	1	70	300,000
Sammi Steel, Changwon	1	30	225,000
	2	15	
	1	3	
Seoul Steel, Inchon	1		60,000
Songjia Works, Songjin			
MALAYSIA			
Amalgamated, Selangor	1	90	200,000/ 300,000
Malayawata, Prai	1	10	24,000
MEXICO			
Aceros Industriales, Mexico City	1		36,000
Atlax SA, Nr. Apizaco	1	60	200,000
Ansa, Tlalnepantla	3		320,000
Laminadora Atzcapotzalco, Mexico City	1		88,000
Campos Hermanos, Tlalnepantla	1	17	34,000
	1	35	41,000
Chihuahua, Chihuahua	3		110,000
Corsa, La Presa	1	12ft dia	100,000
Ecatepec	3	10ft dia	100,000
Hierro y Acero, Mexico DF	1	15	
	2	4	
Guadalajara, Guadalajara	2	25	125,000
	1	55	300,000
Hylsa, Monterrey	1	13ft dia	1,100,000
	2	18ft dia	
	2	18ft dia	
Peubla, Apodaca,	3	17ft dia	600,000
Monterrey	1	10ft dia	130,000
	1	10ft dia	
Omega, Antigua Carretera	2	12	60,000
Potosina, San Luis Potosi	1	30	
San Luis, San Luis Potosi	2	11ft dia	120,000
Sidena, Ciudad Sahagún			
Solar, Tlalnepantla	2	10	7,800
	1	5	1,200
Tamsa, Veracruz	4	50	420,000
Veracruzana, Córdoba	3		80,000
Yucatan, Merida-Progreso	1		110,000
	1		
NETHERLANDS			
Nedstaal, Alblasserdam	2	35	235,000
NEW ZEALAND			
NZ Steel, Glenbrook	2	50	160,000
Pacific Steel,			
NIGERIA			
Delta Steel, Ovwian-Aladja	4	110	1,000,000
Lapec, Epe			
Nigersteel, Enugu			
NORWAY			
Elkem, Oslo	1	60	175,000
Norsk Jernverk, Mo i Rana	1	75	110,000
	2	60	140,000
PAKISTAN			
Karachi Rolling Mills, Karachi	1	6	18,500
	1	5	16,500
Nowshera Eng, Nowshera	2	5	30,000
	2	2.5	
Sind Steel, Karachi	1	6.5	12,500
Special Steels of Pakistan, Karachi	2	10/12	33,000
	1	3/4	
United Steel Re-Rolling, Lahore	2	1	9,000
PERU			
Laminadora del Pacifico, Pisco	2	40	
Siderperú, Chimbote	4		226,000
PHILIPPINES			
Apollo Enterprises, Mandaluyong	1	15	36,000
Armco-Marsteel, Taguig	1	3.9 metres	70,000
Armstrong, Metro Manila	1	3.2 metres	30,000
Marcelo, Sta Ana	1	12	34,000
	1	10	25,000
	1	8	13,500
	2	3	10,000
National Steel, Iligan	1	25	45,000
POLAND			
Huta Baildon, Katowice			
Huta Batory, Chorzow-Batory			
Huta Bierut, Raków and Mirow, Czestochowa,			
Jednosc, Siemianowice	1	70	
Nowotko, Ostrowiec	3	140	1,000,000
Huta Pokój Nowy Bytom			
Huta Warszawa, Mlociny			
Huta Zabrze, Zabrze			
Zawiercie, Zawiercie	4	140	
PORTUGAL			
Siderurgia Nacional, Seixal	1	45	120,000
Maia	1	70	210,000
PUERTO RICO			
Insid, Bayamon	2	20	120,000
QATAR			
Qasco, Doha	2	70	400,000
RUMANIA			
Calarasi, Calarasi	2		
Cîmpia Turzii, Cîmpia Turzii			
Hunedoara, Hunedoara	2	50	
	4	25	
Iași, Iași	1	50	
Oțelul Roșu	1	60	
	1	110	
Tirgoviște, Tirgoviște	6	100, 50, etc	
SAUDI ARABIA			
Hadeed, Al-Jubail	3	125	850,000

Continued overleaf

ELECTRIC ARC FURNACES

	No.	Size (tonnes)	Capacity (tpy)
SINGAPORE			
Nism, Jurong	2	20	100,000
	1	40	160,000
	2	50	340,000
Partap Special Steels, Jurong	2	15/17	50,000
SOUTH AFRICA			
Cape Town Iron & Steel, Kuils River	1	30	120,000
Davsteel, Vanderbijlpark	1	50	
Dunswart, Benoni	1	50	180,000
	2	40	200,000
Iscor, Pretoria	1	60	
	1	15	
	2	125	
Vanderbijlpark	3	150	
McWilaw, Isipingo			
Middelburg, Middelburg	1	50	
	1	25	
Scaw, Germiston	1	3	300,000
	1	10	
	1	12	
	1	15	
	1	20	
	1	50	
Usco, Vaal	3	45	250,000
Klip	2	15	50,000
SPAIN			
Aceriasa, Santander	2		520,000
Acerinox, Algeciras	1	90	200,000
	1	100	100,000
Aceros Corrugados, Lezo	1		
AHV, Baracaldo-Sestao	1	50	120,000
Forjas, Alavesas, Vitoria	3	60	300,000
	1	35	60,000
Aceria de Amurrio Alava	1	90	220,000
Aristrain, Olaberria	2	70	300,000
Aristrain-Madrid, Madrid	1	100	200,000
	2	15	100,000
Arregui SA, Vitoria			
Artierro SA, Silla			
Astilleros Españolas SA, Reinosa			
Azcoitia, Azcoitia	1	8.5	
	1	11.2	
	1	12.3	
	1	30	
Azma, SA, Getafe			
Babcock & Wilcox Española, Galindo	1	100	350,000
	1	50	
Industrias del Besos S. Adrian de Besos	3		127,050
Altos Hornos de Cataluña, Hospitalet de Llobregat	2		
Celsa, San Andres de la Barca	1	80	225,000
Echevarria, Basauri			
	2	40	120,000
	1	80	210,000
Patricio Echeverria	1	40	
	4	20	
Guadiana Villafranca de los Barros			
Inoxca, Barcelona	3		3,000
Irura, Irura			
Jaca, Jaca			
Aceros de Llodio SA, Llodio			
Victoria Luzuriogo SA, Usurbil San Sebastian			
Macosa-Material Construcciones SA, Barcelona			
Megasa, La Coruña	1	35	
	1	20	
Nervacero, Ballonti	1	100	
	1	70	
Olarra, Lujaua			
Esteban Orbegozo, Zumárraga			
Pedro Obegozo, Hernani	1	1	1,400
	1	5	8,000
	1	20	30,000
	1	50	70,000
	2	70	220,000
Rico y Echeverria, Zaragoza			
Roldan, Ponferrada	1	3	6,000
Sevillana, Alcala de Guadaira			
Sidegasa, Teixeiro	1	100	323,000
Torras, Herreria, Poligono San Vicente			
Tubacex			200,000
Marcial Ucin, Azpeitia	1	80	300,000
Unión Cerrajera, Vergara			
SRI LANKA			
Ceylon Steel, Athurugiriya	1	25	65,000
SWEDEN			
Avesta, Avesta	2	50	200,000
Björneborg, Björneborg	1	55	60,000
	1	25	
Bofors-Kilsta, Bofors	1	50	70,000
Fagersta, Fagersta	1	65	110,000
Gullspång, Gullspång	1		25,000
Halmstad, Halmstad	1	50	220,000
Kloster Speedsteel, Söderfors	1	30	
Kockum, Ronneby	1	50	
Nyby Uddeholm AB, Degerfors	1	70	210,000
Sandvik, Sandviken	1	75	120,000
	1	50	60,000
SKF Steel Hällefors, Hällefors	1	70	300,000
SKF Steel Hofors, Hofors	1	80	400,000
Smedjebacken-Boxholm, Smedjebacken	1	125	350,000
SSAB, Domnarvet	2		420,000
Uddeholm Tooling, Hagfors	1	35	35,000
	1	65	70,000
SWITZERLAND			
Ferrowohlen, Wohlen	3		150,000
Monteforno, Bodio	3	80	
Von Roll AG, Gerlafingen	2	50	250,000
SYRIA			
Gecosteel, Hama	2	25	
TAIWAN			
Chia Hsin, Fung Yuen	1		
Dah Yung, Kaohsiung	1	25	86,000
	1	25	72,000
Feng Hsin, Taichung Hsien	2	25	216,000
Hai Kwang, Kaohsiung	1	4.600 mm dia.	100,000
Lien Ho Hsing Kaohsiung	1	15	36,000
Tai Lee Industrial, Taipei	1	20	60,000
Tang Eng, Kaohsiung	2	30	180,000
	1	40	130,000
Tung Ho, Kaohsiung	1	20	
THAILAND			
Bangkok Iron & Steel, Prapradaeng	6		
Bangkok Steel Industry, Samudprakarn	2	20	120,000
GS Steel, Samrongtai Prapradaeng	3	20	158,000
Siam Iron & Steel, Saraburi	2	30	120,000
Thai-India Steel, Samudprakarn	2		45,000
TOGO			
Sté Nationale de Sidérurgie, Lomé	1	12	20,000
TRINIDAD & TOBAGO			
Iscott, Point Lisas	2	90	600,000
TUNISIA			
Elfouladh, Menzel Bourguiba	1	15	30,000
TURKEY			
Asil Celik, Orhangazi-Bursa	1	15	30,900
	1	45	92,500
Colakoğlu Metalurji, Gebze	1	10	400,000
	4	30	
Cukurova Celik, Aliağa	2	75	300,000
Elmet-Elektro-Metal Sanayii, Eyüp-Istanbul	2	9	
Içdaş, Sinnevler-Bakirköy	1	25	100,000
	1	15	
IDC, Izmir	2	50§	
Kroman, Cayirova-Gebze	1	15	30,000
	1	15	20,000
Metaş, Izmir	1	10	20,000
	2	15	60,000
	2	45	280,000
MKEK, Kirikkale			
Tuber-Celik Sanayii, Istanbul	1	45	
UK			
Allied Steel & Wire, Cardiff	2	100	500,000
Alphasteel, Newport	4	100	1,500,000
Brymbo, Brymbo	1		560,000
BSC, Clydesdale	2	70	350,000
Craigneuk	1	25	45,000
Hamilton, Merthyr Tydfil	1	3	1,000
Panteg	1	44	75,000
Rotherham	1	120	2,500,000
	6	175	
Shepcote Lane and Tinsley Park Works, Sheffield	1	110	300,000
Stocksbridge and Tinsley Park Works, Sheffield	2	150	980,000
	2	135	
	1	120	
	2	15	
C. G. Carlisle, Rotherham	1	6	
Hadfields, Sheffield	1	25	
	1	80	
F. H. Lloyd, Wednesbury	1	35	70,000
	1	25	30,000
Lloyds (Dudley), Dudley	1	35	70,000
Manchester Steel, Manchester	1	50	130,000
Bidston	1	55	140,000
Sanderson Kayser, Sheffield	1	10	
	1	15	
Sheerness, Sheerness	2	100	450,000
Sheffield Forgemasters, Sheffield	1	90	100,000
Spear & Jackson, Sheffield	1	7.5	5,000
Unbrako, Kiveton Park	1	20	30,000
Weardale Steel, Wolsingham	1		30/35
			21,000
	2	4/5	4,000
URUGUAY			
Inlasa, Montevideo	1	30	
Pando Industrial Zone	1	30	
Rattur, Montevideo	1	10	
USA*			
Al Tech, Watervliet and Dunkirk, NY	2	20	
Allegheny Ludlum Brackenbridge, PA	2	60	
	2	70	

	No.	Size (tonnes)	Capacity (tpy)
Alloy Metal Products, Davenport, IA	1	3	
	1	3	
Ameri-Steel, Uniontown, PA	1	50§	
Armco, Butler, PA	3	165	
Kansas City, MO	2	100	
	4	150	
Baltimore, MD	1	50	
Atlantic Steel, Atlanta, GA	2	85	450,000
Cartersville, GA	1	100	250,000
Auburn, Auburn, NY	1	16ft	240,000
Babcock & Wilcox, Beaver Falls, PA	2	25	
	2	50	
	2	75	
	3	100	
Bayou, LaPlace, LA	2	65	700,000
Bethlehem, Bethlehem, PA	1	28	
	4	50	
Johnstown, PA	2	185	
Steelton, PA	1	5	
	1	15	
	3	150	
Seattle Division, Seattle, WA	2	120	
Birmingham Bolt, Birmingham, AL	2	25	
Bourbonnais, IL	2	15	100,000
Border Steel Mills, Vinton, TX	2	20	200,000
Braeburn Alloy Steel, Braeburn, PA	2	10	24,000
Calumet, Chicago Heights, IL	2	30	150,000
Cameron Iron Works, Houston, TX			
Carpenter Technology, Reading, PA	6		
Bridgeport, CT	2		
Cascade, McMinnville, OR	2		
CF & I, Pueblo, CO	2	150	
Champion, Orwell, OH	1	8.5ft dia	
			4,875
Chaparral, Midlothian, TX	2	130	900,000
Columbia Tool Steel, Chicago Heights, IL	1	5	
	1	8	
Connors, Birmingham, AL**	2	40	270,000
Continental, Kokomo, IN	2	175	
Copperweld, Warren, OH	5	90	750,000
Crucible, Syracuse, NY	1	35	
	3	15	
Cyclops, Mansfield, OH	2	100	
Bridgeville, PA	3		
Eastern Stainless, Baltimore, MD	1	50	100,000
Edgewater, Oakmont, PA	1		
Electralloy, Oil City, PA	1	35	60,000
Finkl, Chicago, IL	2	65	100,000
Florida, Charlotte, NC	2	35	200,000
Indiantown, FL	1	35	92,000
	1	30	78,000
Baldwin, FL	1	85	400,000
Tampa, FL	2	35	200,000
Jackson, TN	1	135	400,000
Georgetown, Georgetown, SC	2	75	
Green River, Owensboro, KY	2	65	
Hawaiian Western, Ewa, HI	1	15/20	60,000
Hurricane, Sealy, TX	1	16ft	90,000
Ingersoll Johnson, Newcastle, IN	2	3	
	2	12	
Ingersoll-Rand, Pampa, TX	1	22	25,000
Inland, Indiana Harbor, IN	2	120	500,000
Inmetco, Ellwood City, PA	1	7 metre dia	30,000
Intercoastal, Chesapeake, VA	2	11ft dia	100,000
Jessop, Washington, PA	3	20	60,000
Earle M. Jorgensen, Seattle, WA	2	40	
Joslyn Stainless, Fort Wayne, IN	3	15	
Judson, Emeryville, CA	1	60	
Kentucky Electric, Coalton, KY	2	45	250,000
Keystone, Peoria, IL	2	175	650,000
Knoxville, Knoxville, TN	2	30	168,000
Laclede, Alton, IL	2	225	840,000
Latrobe, Latrobe, PA		30	
Lone Star, Lone Star, TX	2	60	
LTV Steel, Pittsburgh, PA	2	350	
Midland, PA	2	170	
Canton, OH	3	90	
	4	200	
Chicago, IL	3	200	
Lukens, Coatesville, PA	2	150	
	2	100	
Marathon LeTourneau, Longview, TX	2	25	
Marathon, Tempe, AZ	3	25	10,000
Marion, Marion, OH	2	30-40	
McLouth, Trenton, MI	2	200	350,000
Mississippi, Flowood, MI	1	12ft 6in	
National Forge, Irvine and Erie, PA	2	75	
	1	40	
	1	35	
	1	15	
National Steel, Detroit, MI	2	185	
National Supply, Torrance, CA	1	22.5	
	1	10	
New Jersey, Sayreville, NJ	2	70	240,000
Newport Steel, Newport, KY	3	100	510,000
North Star, St Paul, MN	2	75	450,000
Monroe, MI	1	120	600,000
Wilton, IA	1	80	300,000
North Star Texas, Beaumont, TX	2	120	
Northwestern Steel & Wire, Sterling, IL	3	400	2,400,000
Nucor, Darlington, SC	3	32	250,000
	2	32	275,000
Norfolk, NE	2	50	350,000
	1	35	
	2	50	225,000
Jewett, TX	2	50	250,000
	3	50	300,000
Plymouth, UT	2	60	450,000
Oregon, Portland, OR	2	80	500,000
Owen Electric, Cayce, SC			100,000
Quanex, Jackson, MI	2	40	
Raritan River, Perth Amboy, NJ	1	20ft dia	600,000
Razorback, Newport, AR	2	25	160,000
Rhode Island Forging Steel, Rumford, RI	2	40	
Roanoke, Roanoke, VA	1	18ft	500,000
	1	14ft	
	2	11ft	
Roblin, Dunkirk, NY	2	25	100,000
Rouge, Dearborn, MI	2	220	800,000
Sharon, Farrell, PA	2	120	
Sheffield Steel, Sand Springs, OK	2	75	
Soulé, Carson, CA	2	11ft dia	120,000
Standard Steel, Burnham, PA	1	40	62,208
	1	45	65,664
	1	70	110,592
	1	18	24,192
Latrobe, PA	1	35	65,000
Steel of West Virginia, Huntington, WV	2	70	
Structural Metals, Seguin, TX	1	90	350,000
Tamco, Etiwanda, CA	1	20ft dia	300,000
Teledyne Vasco, Latrobe, PA	1	18	
Tennessee Forging, Harriman, TN	3	25	
Texas Steel, Fort Worth, TX	1	30	96,000
	1	25	80,000
	1	13	16,000
	1	1	8,000
Thomas Steel, Lemont, IL	2	40	
Timken, Canton, OH	3	150	1,125,000
	4	100	
	1	60	
Union Electric, Burgettstown, PA	1	14ft	50,000
US Steel, Fairless, PA	2	200	
Baytown, TX	2	200	
	2	220	
Duquesne, PA	1	20	
	1	70	
	3	85	
South Chicago, IL	2	200	
	1	100	
USS Specialty Steel Products, Johnstown, PA	1	30	
	2	3	
Washington, Houston, PA	2	45	140,000
USSR			
Amur, Komsomolsk-on-Amur			
Azovstal, Zhdanov			700,000
Beloretsk, Beloretsk	1		
Chelyabinsk, Chelyabinsk	12		
Dnieper Special Steel, Zaporozhye	20		
Donetsk, Donetsk		100	
Elektrostal (Novokramatorsk), Kramatorsk			
Elektrostal Metallurgechesky Zavod Imeni Tevosyan			
Izhevsk, Izhevsk	7		
Kirov, Makeyevka	1		
Kirov Works, Leningrad	3		
Kursk, Stary Oskol	2	150	
Novokuznetsk, Novokuznetsk	5		1,250,000
Novolipetsk, Lipetsk	2	100	
Orsk-Khalilovo, Novo Troitsk	2		
Red October, Volgograd	1	200	
Serov, Serov			
Sickle & Hammer, Moscow	4	10	
Taganrog, Taganrog	2		
Uzbek, Bakabad	3		750,000
Verkh-Isetk, Sverdlovsk	3		
Western Siberian, Novokuznetsk	1		
Zhlobin Metallurgical Works, Zhlobin	2§	100	
Zlatoust, Zlatoust			
VENEZUELA			
Sidetur, Barquisimeto	1	30	60,000
Sidor, Matanzas	6	200	2,400,000
Sivensa, Caracas	2	23	170,000
	1	6	25,000
YUGOSLAVIA			
Boris Kidrič, Nikšić	2	60	400,000
	1	25	
	3	10	
RMK-Zenica, Zenica	1	10	
	1	3	
Sisak, Sisak	1	30	70,000
Skopje, Skopje	1	130	
Slovenian Steelworks,	2		180,000
Ravne	2	40	110,000
	1	30	50,000
	1	10	20,000
Store	1	40	60,000
	1	60	80,000
Split, Split	1	20/23	56,000
	1	25	65,000
ZAIRE			
Sté Nationale de Sidérurgie, Maluku	1	55	120,000

**short tons. §under construction. **not in operation.*

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APPENDIX 5

Maximum Impurity Levels by Grade

APPENDIX 5

MAX IMPURITY LEVELS - STEEL PRODUCERS

Typical Residual Levels - Scrap Steel

Element \ Class	0	I	II	III	IV	V
Cu (% max.)	0.005	0.010	0.100	0.150	0.200	0.300
Sn	0.001	0.010	0.010	0.015	0.020	0.030
Ni	0.005	0.010*	0.100*	0.100*	0.100	not specified
Cr	0.020	0.050*	0.050*	0.050*	0.050	not specified
Mo	0.002	0.010*	0.020*	0.020*	0.020	not specified
S	0.002	0.010	0.015	0.020	0.035	0.040
P	0.005	0.010	0.015	0.030	0.040	0.050
Steel Products	aviation grade; nuclear grade; special petroleum grade; electroslag refining; special bearing grade	maraging; extra deep drawing; high strength; fine drawing wire; special alloy; bearing; tinplate for drawing and iron cans	deep drawing; cold drawing, forming, heading; stainless steel; alloy steel; c.c. rounds for seamless tubes	forging bars; auto parts; punching strips; alloy steel for heat treating; pressure vessels	standard tube, pipe; tinplate; black plate; wheels; axles; cold finished wire rod; galvanized and enameling sheet	wire rod; coarse round wire; rebar; angles; shapes; sections

*for carbon steel grades only

Table I General Tramp Element Limits for Steel Products

Element	No. 1 Bundles	Shredded	No. 1 H.M.	No. 2 H.M.	No. 2 Bundles	HBI
Cu	0.10	0.20	0.20	0.40	0.60	0.01
Sn	0.01	0.02	0.02	0.03	0.05	0.00
Ni	0.05	0.10	0.10	0.13	0.14	0.01
Cr	0.09	0.12	0.15	0.14	0.18	0.01
Mo	0.01	0.01	0.02	0.02	0.02	0.00
S	0.02	0.04	0.04	0.07	0.09	0.01
P	0.01	0.02	0.02	0.03	0.03	*

*Subject to phosphorus content in the ore

Source - "Direct from Midrex" v. 10, No. 3, 2nd Q, 1986

APPENDIX 6

Consumption of Scrap, 1985 - IISI

APPENDIX 6

CONSUMPTION OF SCRAP

1985

Million Tonnes

Belgium/Luxemburg	4.6
Denmark	0.6
France	6.4
West Germany	18.6
Italy	15.5
Netherlands	1.5
Portugal	0.1 E
Spain	10.6 E
U.K.	8.4
TOTAL ABOVE	66.5
Austria	1.5
Finalnd	0.8 E
Norway	0.6 E
Turkey	1.8
Yugoslavia	2.3
TOTAL WESTERN EUROPE	76.6
Canada	7.2
U.S.A.	48.5 (a)
Japan	44.2
TOTAL ABOVE	176.5
USSR & Eastern Block	79.8
TOTAL	256.3

(a) blast furnace and steelmaking only

Source IISI

APPENDIX 7

Comparative Prices - Iron Ore and Steel Products

APPENDIX 7

COMPARATIVE TYPICAL SELLING PRICES*

(1)	Sinter Feed	26.50¢ per metric ton unit
	Iron Ore Pellets (FOB Port Cartier)	36.50¢ per metric ton unit
	Hot Metal	105 (est cost) net ton
	Liquid Steel	140 (est cost) net ton
(3)	Billets (Merchant)	155 net ton
(3)	Billets (SBQ)	275 net ton
(3)	Rebar	240 net ton
(3)	Bars (Merchant)	280 net ton
(3)	Bars SBQ (Hot Forging)	330 net ton
(3)	Wire Rod	315 net ton
(3)	Slab (Cast)	270 net ton
(3)	Hot Band (Commercial)	340 net ton
(3)	Hot Rolled (Seconds)	200 net ton
(3)	Cold Rolled Coil (Commerical)	460 net ton
	(Seconds)	250 net ton

*1987 US dollars except where noted

Sources:

(1) IISI
(2) Skillings
(3) Battle for the U.S. Market - D.F. Barnett (1987)

APPENDIX 8

Steel Statistics - Developing Countries

APPENDIX 8

Steel Statistics - Developing Countries

	Imports Semi-Fin. & Fin. Steel Prod. 000 tonnes		Crude Steel Production 000 tonnes		Apparent Consumption - Crude Steel Per Capita kg.		GNP Per Capita U.S. $
	1985	1986	1985	1986	1985	1986	1985
China	19,235	18,470	46,724	51,900	67	70	310
Indonesia	931	1,250 E	1,200	1,500 E	N.A.	N.A.	530
Malaysia	1,500	1,200 E	550	750 E	160	143	2,050
Philippines	346	549	250	250 E	N.A.	N.A.	600
Singapore	1,695	1,710	365	390	N.A.	N.A.	7,420
Taiwan (ROC)	1,258	1,910	5,088	5,545	N.A.	N.A.	-
Thailand	1,093	1,500 E	330	3,887 E	N.A.	N.A.	830
Argentina	465	561	2,946	3,235	69	78	2,130
Chile	76	138	689	706	47	51	1,440
Columbia	604	600 E	530	632	46	48	1,320
Ecuador	344	300 E	18	17	N.A.	N.A.	1,160
El Salvador	45	50 E	N.A.	N.A.	N.A.	N.A.	710
Guatemala	101	100 E	N.A.	N.A.	N.A.	N.A.	1,240
Honduras	58	55	N.A.	N.A.	N.A.	N.A.	730
Paraguay	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	940
Peru	135	59	397	486	28	28	960
Uruguay	N.A.	N.A.	39	31	N.A.	N.A.	1,660
Algeria	505	899	800	800 E	91	88	2,530
Kenya	215	149	N.A.	N.A.	N.A.	N.A.	290
Morocco	642	453	N.A.	N.A.	38	26	610
Nigeria	2,148	1,489	321	200	33	22	760
Sri Lanka	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	370
Zaire	24	19	N.A.	N.A.	N.A.	N.A.	170
Zimbabwe	109	113	731	674	N.A.	N.A.	650
Iraq	852	520	N.A.	N.A.	69	41	N.A.
Libya	488	N.A.	N.A.	N.A.	176	155	7,500
Saudi Arabia	3,450	1,487	1,106	1,100 E	484	252	8,860
Egypt	1,900	N.A.	950	1,000 E	70	49	680
Greece	1,196	1,266	985	1,010	163	156	3,550
Turkey	3,070	2,155	4,864	2,928	100	995	1,130

E = ESTIMATE

PROJECT REPORT

HATCH ASSOCIATES LTD. CONSULTING ENGINEERS

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STELCO - LB PROCESS EVALUATION

December 11, 1987

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SUMMARY REVIEW OF WORLD ENERGY TRENDS

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BIBLIOGRAPHY

APPENDICES

S. Kumar

S. Kumar

If you disagree with any information contained herein, please advise immediately.

1.0 INTRODUCTION

A survey has been carried out of World Energy Trends (demand, production, reserves and prices) as they relate to the LB process evaluation. The rationale for this study is that since the LB process is coal based, it would be more suitable, with all other factors constant, in an area of relatively cheap coal.

The extent of this study reflects the accuracy and quantity of information required at this stage of the evaluation. A full list of all data sources contacted is included.

2.0 NATURAL GAS AND COAL RESERVES

The two most critical energy/reductant forms in the production of iron via the blast furnaces, DRI or LB routes, are natural gas and coal; therefore they are discussed in greater detail.

Table 1 summarizes the estimated economically recoverable natural gas reserves by geography. The largest reserves are in USSR-Eastern Europe (44% of world total) and Middle-East (25% of world total). If the world's natural gas supply was consumed at present rates, this supply would last about sixty years.

Table 1 also gives the estimated economically recoverable coal reserves by geography and coal type. The largest reserves are in North America (predominantly USA) at 30% of world total, USSR-E. Europe at 33% and China at 11%. If the world's total coal supply was consumed at present rates, at least two hundred years supply would be available.

It should be borne in mind that the recoverable reserves increase both as geological exploration proceeds and as the technology of recovery proceees are improved.

More detailed data is included in Appendix A.

TABLE 1: SUMMARY OF WORLD RECOVERABLE COAL & NATURAL GAS RESERVES

REGION	Estimated natural gas reserves (trillion cf)(2)	Anthracite + Bituminous Coal		Recoverable Lignite (billion TON)	Total Recoverable Coal (billion TON)
		Total Recoverable (billion TON)	Coking Quality (billion TON)		
N. America	370.1	254.31	2.75	37.58	291.90
C. & S. America	112.4	17.28	1.37	0.02	17.30
W. Europe	226.4	43.26	23.04	59.31	102.58
E. Europe - USSR	1516.0	199.33	66.02	122.96	322.89
Africa	198.4	65.27	0.15	0.11	65.38
M. East	855.8	0	0	0	0
F. East + Oceania	201.5	144.44	53.99	42.23	186.67
(China) (4)	(30.0)	(108.90)	(40.29)	(0)	(108.90)
TOTAL	3480.6	723.89	147.31	262.21	986.72

NOTES:
1) SOURCE - International Energy Annual 1985, by U.S.D.O.E.
2) Natural gas reserves are as of Jan/86.
3) Coal reserves are based on 1981 data.
4) China is included in F. East figures.
5) Ton is short ton.

3.0 HISTORICAL NATURAL GAS AND COAL PRODUCTION

The annual production of dry natural gas and coal by region are summarized in Figures 1 and 2. The two major natural gas producing areas are North America and USSR - E. Europe (predominantly USA and USSR). For coal, the major regions are North America, USSR - E. Europe and Far East; with USA, USSR and China being the dominant producers in this group. More detailed data is given in Appendix B.

Table 2 is a breakdown of 1984 world coal production by coal type as well as region. On a total world tonnage basis, bituminous coal accounts for about two-thirds of the production, and lignite coal about 28%. The largest producers of metallurgical coke are USSR, Japan, China and U.S.

WORLD DRY NATURAL GAS PRODUCTION

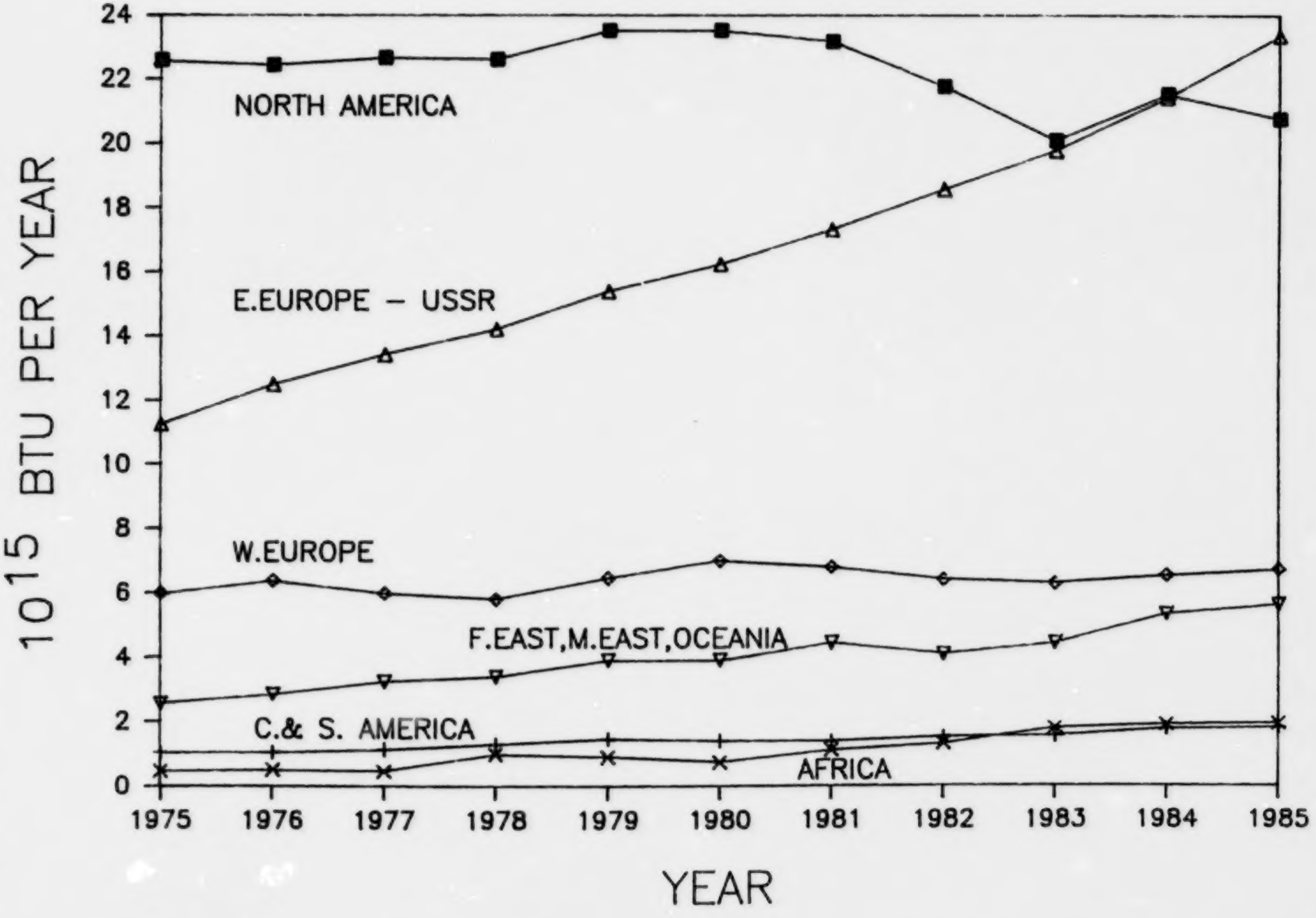


FIGURE 1

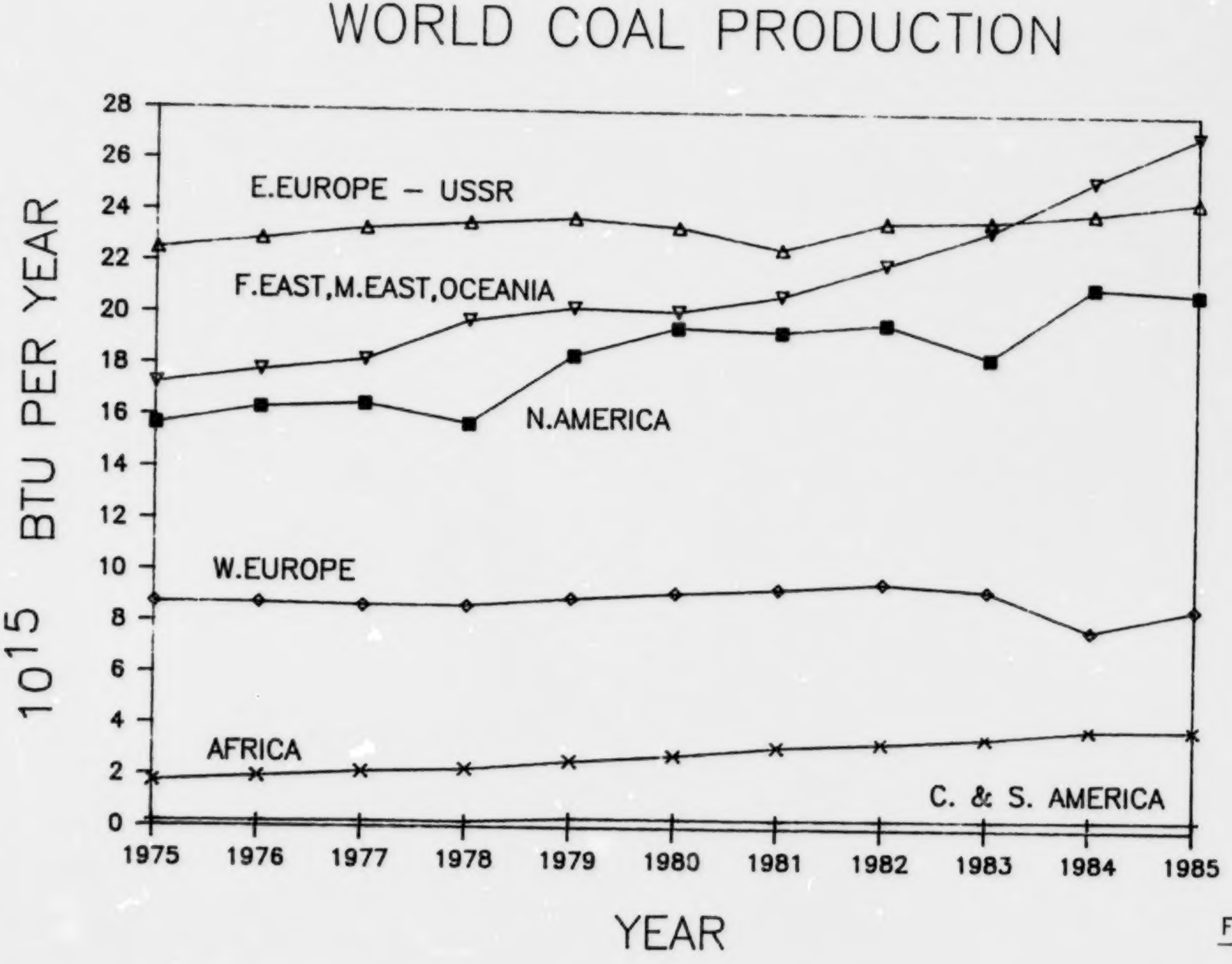

WORLD COAL PRODUCTION
10^{15} BTU PER YEAR
28
26
24
22
20
18
16
14
12
10
8
6
4
2
0
E.EUROPE – USSR
F.EAST,M.EAST,OCEANIA
N.AMERICA
W.EUROPE
AFRICA
C. & S. AMERICA
1975
1976
1977
1978
1979
1980
1981
1982
1983
1984
1985
YEAR

FIGURE 2

TABLE 2: WORLD 1984 COAL PRODUCTION BY TYPE

Region	Primary Coal (Thousands Ton) Anthr.	Bit.	Lignite	Metallurgical Coke (2) (Thousands Ton)
N. America	4,162	890,951	74,002	38,438
C. & S. America	140	17,371	45	8,143
W. Europe	16,977	182,519	310,039	73,016
E. Europe - USSR	93	866,066	768,946	133,381
Africa	5,318	178,667	0	3,344
F. East + Oceania	238,275	978,301	128,541	121,884
(China) (3)	(170,000)	(666,785)	(33,190)	(39,848)
World Total	264,965	3,114,875	1,281,573	378,591

NOTES: 1) Source - International Energy Annual 1985, by U.S.D.O.E.
2) Metallurgical Coke is derived from Primary Coal.
3) China is included in F.East.

4.0 ENERGY PRODUCTION FORECASTS

Analysis by the World Energy Conference (see Bibliography item (V)) has led to the following forecasts for world energy production:

	% Primary Energy Production		
	1978	2000	2020
Oil	42	32	22
Natural Gas	16	28	18
Coal	24	27	30
Other Primary Energy Forms	18	23	30
Total (%)	100	100	100
Total (million tonnes oil equivalent)	7,179	12,384	18,534

It can be seen that the relative importance of coal in the world energy situation will increase. It is also to be noted that nuclear and coal energy do not represent suitable direct substitutes in some important applications, such as transportation and feedstocks, of hydrocarbon fuels.

The production forecasts breakdown by geography is given in Table 3. Major observations are as follows:

Energy Form	Major Production Trends Forecast
Coal	Increase in North America, USSR - E. Europe South Africa, China
Oil	Increase in Latin America, China
Natural Gas	Increase in USSR-E. Europe, North Africa and Middle-East

TABLE 3: FORECASTS OF ENERGY PRODUCTION

In million tonnes oil equivalent (1 toe = 44 x 10^9 Joules)

	SOLID MINERAL FUELS			OIL			NATURAL GAS		
	1978	2000	2020	1978	2000	2020	1978	2000	2020
1. U.S. & Canada	380	1067	1958	574	570	484	527	502	460
2. W. Europe	220	305	360	92	158	115	152	121	103
3. Australia, Japan, New Zealand	71	180	290	25	35	50	10	30	60
4. E. Europe (incl. USSR)	599	790	1000	575	595	520	344	835	1200
5. N. Africa, Middle East	1	5	8	1182	1340	1310	59	261	515
6. Sub-Saharan Africa	53	188	468	113	180	250	1	10	30
7. South Asia	49	132	254	12	55	85	8	40	80
8. S.E. Asia (incl. Korea)	12	70	130	101	150	120	18	100	200
9. Communist Asia (incl. China, N.Korea)	307	530	780	104	315	400	12	85	300
10. Latin & S. America	8	55	290	240	541	822	52	189	335
TOTAL	1,700	3,322	5,538	3,018	3,939	4,156	1,183	2,173	3,283

NOTES: 1) Solid Mineral Fuels is predominately coal.

5.0 ENERGY PRICES

5.1 Available Data

Extensive information on energy prices is not readily available for all regions of the world. Data on the OECD countries (essentially the industrialized nations) is well compiled; by comparison, data on developing countries is sparse.

Caution is required in comparing data collected by different sources, as they may not be on the same basis.

5.2 Current Prices

A comparison of prices of different types of energy for selective countries is given in Table 4. The oil price has been taken to be the same for all the countries. The cost of coal in all the listed countries is generally the same, with the exception of Japan, where it is more expensive. Electricity is most expensive in Japan and U.S.

Table 4 also gives a relative comparison of some International electricity prices for large industrial users (with Ontario being the base) in 1985. The indices range from 79 (percent of Ontario rate) in South Africa to nearly 400 in the U.S.

The availability of abundant low-cost oil in 1960's led most countries to substitute oil for coal as the primary fuel for industry and utility boilers. There was a renewed interest in coal in 1970's as the oil prices increased. Currently, the low oil prices have depressed coal prices in the spot market as utilities use greater quantities of oil.

5.3 Price Forecasts

The range of world crude oil price forecasts by three different groups are summarized in Figure 3. The general trend is predicted to be the same, but prices differ. The 1986 price was about US$15 per barrel, the forecast is for a $22 to $27 range in 1995, $32 to $40 range in 2000, and $40 to $47 range in 2010 (all in 1986 dollars).

TABLE 4: CURRENT INTERNATIONAL ENERGY PRICES

A) INDUSTRIAL PRICES FOR DIFFERENT ENERGY TYPES IN SOME COUNTRIES

All costs in 1986 U.S. Dollars

COUNTRY	OIL ($/MMBTU)	GAS ($/MMBTU)	COAL ($/MMBTU)	ELECTRICITY ($/kWh)
Australia	3.10	1.55 - 3.70	1.13	0.019 - 0.042
Brazil	3.10	-	1.31	0.030
Japan	3.10	-	2.54	0.069 - 0.093
U.S.S.R.	3.10	2.40	1.51	0.037
U.S.	3.10	2.63	1.44	0.050

Source: Nickdata Inc. publications.

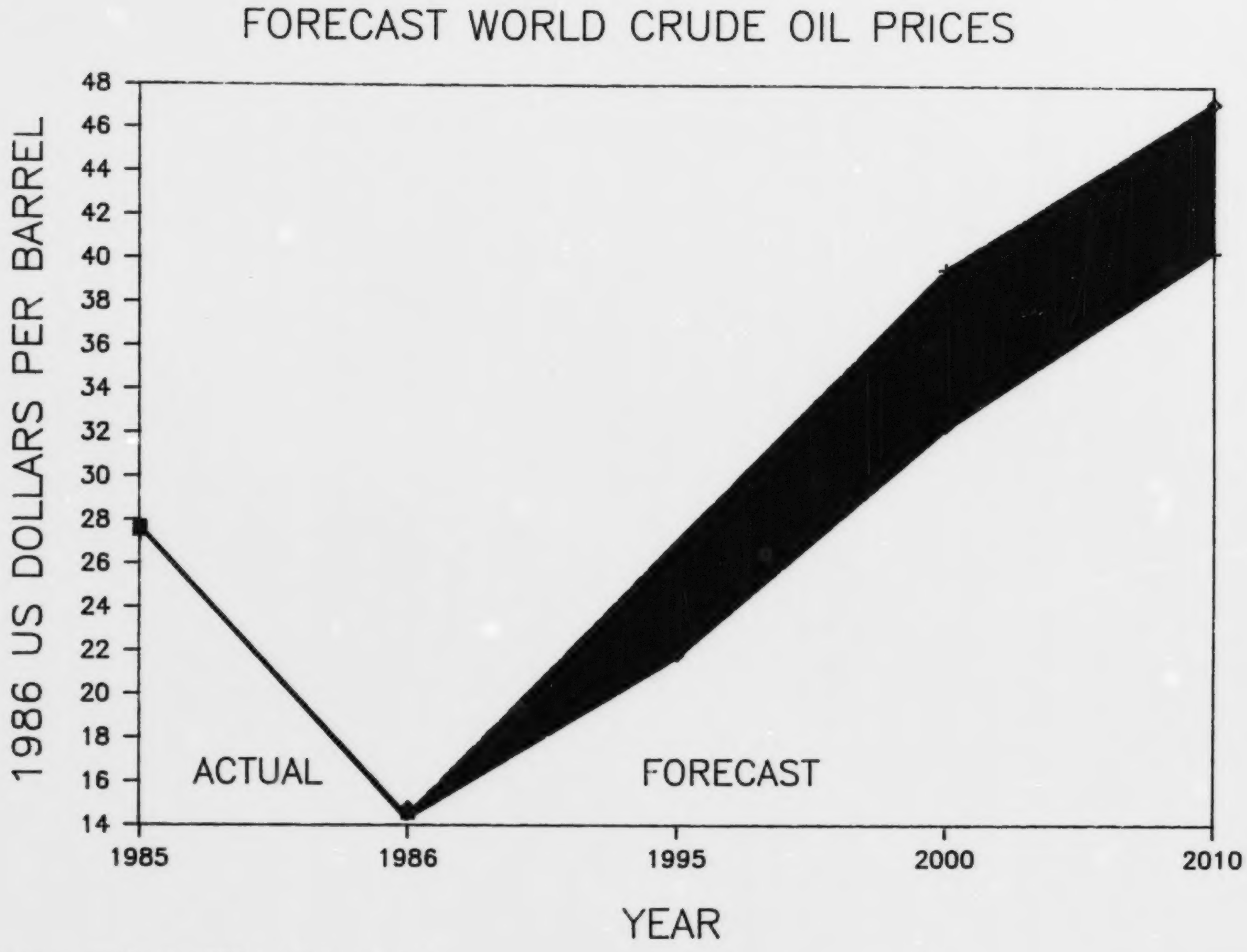
B) RELATIVE COMPARISON OF ELECTRICITY RATES (1985)

Based on service above 10 MW, 80% load factor.

Region	Index
Ontario	100
U.S.	245-385
Australia	117-177
Japan	224-262
Korea	274
England	205
S. Africa	79

Source: Ontario Hydro.

FIGURE 3



SOURCES:

a) Annual energy outlook 1986 by U.S.D.O.E.

b) DRI Energy Review 1987.

c) 1986 GRI Baseline Forecast.

Price forecasts for different types of energy for different geographical regions were not available on a consistent basis. The most extensive data was for the U.S. and is summarized in Table 5. The largest relative price increases from 1986 are forecast to be for oil, and natural gas, whereas coal and electricity prices will be relatively stable. On a $/BTU input basis, coal is and will remain the cheapest energy form in the U.S. Detailed data is in Appendix C.

6.0 WORLD COAL TRADE

Table 6 gives the World Hard Coal Trade in 1985 and 1986. All other factors being equal, a country with significant coal exports may be a more favourable site for the LB process. The total trade in 1986 was 335.7 million tonnes, with a total world hard coal production of 3180 million tonnes. This trade in coal represents about 11% of production whereas for crude oil, trade is over 35% of production.

The major exporters were Australia, U.S., South Africa, Poland. Steam coal shipments accounted for about 48% of total hard coal trade in 1986 and 1985. South Africa has plans to increase production from 181 million tonnes in 1986 to 230 million tonnes in the 1990's with a large part of the increased production going to export. China and Colombia also expect to significantly raise their exports.

If the transportation costs for coal are low, it may be economical for a plant to import coal from another country for use in iron/steel making. Apparent transport costs for coal in 1986 were:

	Steam Coal	Coking Coal
Exported from U.S. to Japan	U $10.85/tonne	U $10.11
Exported from U.S. to Europe	8.53	7.24
Exported from Australia to Japan	11.18	8.03
Exported from Australia to Europe	15.12	15.47

Typical steam coal export prices from U.S. and Australia: U $30-50/tonne (FOB).

TABLE 5: U.S. ENERGY PRICE FORECASTS

All prices in 1986 U.S. Dollars per million BTU.

Energy Type	1985	1986	1995	2000	2010
Residual Oil	3.98	2.17	3.29- 4.40	4.60- 5.38	7.16- 7.35
Natural Gas (Industrial)	3.96	3.07	3.36- 5.31	4.76- 7.03	6.75- 7.40
Electricity (Industrial)	18.95	17.92	12.51-15.68	13.05-16.36	15.17-17.47
Bituminous Coal (Industrial)	1.73	1.71	1.42- 2.01	1.47- 2.14	1.58- 2.35
Metallurgical Coal	2.08	2.07	2.38	2.45	--

Notes:
1) Figures for 1985 & 1986 are actual.
2) Forecasts are a compilation of forecasts by GRI (see Bibliography item vi), DRI (iv) & U.S.D.O.E. (i)

TABLE 6

World Hard Coal Trade in 1985 and 1986

(million metric tons)

Importers / Exporters	N. America 1985	N. America 1986	OECD Europe 1985	OECD Europe 1986	Japan 1985	Japan 1986	Latin America 1985	Latin America 1986	Asia[1] 1985	Asia[1] 1986	Africa + ME 1985	Africa + ME 1986	CPE's 1985	CPE's 1986	Balancing Items 1985	Balancing Items 1986	World 1985	World 1986
Canada	0.4	0.4	2.5	2.4	18.0	17.3	1.2	1.4	4.8	4.2	-	-	0.1	0.1	0.4	0.1	27.4	25.9
U.S.A	14.6	13.3	39.2	39.1	13.8	12.0	6.6	6.3	5.9	7.0	1.9	2.3	2.3	1.9	-0.3	-4.4	84.0	77.5
Australia	-	-	20.5	23.2	44.2	42.2	1.0	1.1	20.2	21.9	1.6	2.0	0.6	1.0	0.5	0.5	88.6	91.9
Other OECD	-	-	14.1	11.6	0.4	-	-	-	0.2	-	0.2	0.1	1.4	0.6	-0.1	2.2	16.2	14.5
OECD	15.0	13.7	76.3	76.3	76.4	71.5	8.8	8.8	31.1	33.1	3.7	4.4	4.4	3.6	0.5	-1.6	216.2	209.8
Poland	-	-	18.8	14.1	-	-	2.1	2.6	-	0.3	0.1	0.1	15.0	17.3	-	-	36.0	34.4
U.S.S.R.	-	-	3.2	4.4	3.8	5.1	-	-	-	-	-	-	16.9	15.9	-	-	23.9	25.4
China	-	-	0.2	0.7	3.6	3.5	-	-	1.1	2.8	-	-	2.9	2.9	-	-	7.8	9.9
Colombia	0.5	0.7	1.6	3.8	-	-	1.4	0.7	-	0.3	0.1	-	-	-	-	-	3.6	5.5
South Africa	0.8	0.9	26.3	22.4	8.7	8.8	-	-	6.3	11.3	2.2	2.1	-	-	-	-	44.3	45.5
Other Non-OECD	-	-	1.7	1.5	0.9	1.1	-	-	0.8	1.3	-	-	1.0	1.3	-	-	4.4	5.2
TOTAL	16.3	15.3	128.1	123.2	93.4	90.0	12.3	12.1	39.3	49.1	6.1	6.6	40.2	41.0	0.5	-1.6	336.2	335.7

Sources: IEA OECD Coal Statistics and other Secretariat sources, including international sources

1. Excludes Asian CPE's and Japan.

Note: Data in the columns N. America to Japan are import statistics. The rows Canada to OECD (except for above mentioned columns) are export statistics. All other data are based on national and international sources and estimates. The sum of coking coal and steam coal may not add up to total hard coal due to rounding, which may also cause discrepancies among tables 4.1 to 4.3 and other trade data in this publication.

7.0 CONCLUSIONS

(1) Total world reserves of coal (on a BTU basis) are over twice that of natural gas and oil combined.

(2) In 1986, the major coal exporting nations were U.S., Australia, South Africa, Poland. China is expected to significantly increase its exports in the future.

(3) Coal is forecast to occupy a greater share of world primary energy production, from 24% in 1978 to 27% in 2000 and 30% in 2020.

(4) Compared to prices for natural gas and oil, coal prices in U.S. will increase at a much lower rate. This is likely true for most other countries.

(5) The relative dominance of coal vs natural gas (based on 1985 energy production) in selected significant steelmaking countries is as follows:

Brazil - Coal
China - Coal
India - Coal
Mexico - Natural Gas
M. East - Natural Gas
S. Africa - Coal
N. & S. Korea - Coal
U.S. - About Equal
U.S.S.R. - Natural Gas

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The following is a list of major organizations contacted or publications reviewed.

(a) Organizations Contacted

Canadian Carbonization Research Association
Canadian Electrical Association
Canadian Energy Research Institute
Canadian National Energy Board
Electrical and Electronic Manufacturers Association of Canada
Energy, Mines and Resources Canada
Eneroptions Canada
Gas Research Institute
Ontario Hydro
Ontario Ministry of Energy
United Nations Statistics Office
United States Department of Energy
World Energy Conference (Ottawa Office)

(b) Publications

i) "Annual Energy Outlook 1986" by U.S.D.O.E. (EIA), February 1987

ii) "Coal Information 1987", by Intl. Energy Agency (OECD), 1987

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iv) "DRI Energy Review", Summer 1987

v) "Energy 2000-2020: World Prospects and Regional Stresses", by J.R. Frisch, World Energy Conference, 1983

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vi) "1986 GRI Baseline Projection of U.S. Energy Supply and Demand to 2010" by P.D. Holtberg et al, Gas Research Institute, December 1986

vii) "International Energy Annual 1985", by U.S.D.O.E. (EIA), October 1986

viii) "International Energy Outlook 1986", by U.S.D.O.E. (EIA), April 1987

ix) Numerous Publications by Ontario Hydro

x) Oil and Gas Journal

xi) "Statistical Review of Coal in Canada, 1986", EMR Canada, 1987

APPENDICES

APPENDIX A: WORLD NATURAL GAS & COAL RESERVES

APPENDIX B: WORLD NATURAL GAS & COAL PRODUCTION

APPENDIX C: ENERGY PRICES IN SELECTED OECD COUNTRIES

APPENDIX A

WORLD NATURAL GAS & COAL RESERVES

Table 24. World Crude Oil and Natural Gas Reserves, January 1, 1986

Region Country	Crude Oil (Billion Barrels)	Natural Gas (Trillion Cubic Feet)
NORTH AMERICA		
Canada	6.5	99.7
Mexico	49.3	77.0
United States	28.4	193.4
Total	84.2	370.1
CENTRAL AND SOUTH AMERICA		
Argentina	2.3	23.6
Bolivia	.2	4.7
Brazil	2.1	3.1
Colombia	1.2	4.1
Ecuador	1.7	4.1
Trinidad and Tobago	.5	10.5
Venezuela	25.6	59.1
Other	1.3	3.2
Total	34.9	112.4
WESTERN EUROPE		
Denmark	.5	3.9
Germany, West	.3	6.4
Italy	.7	5.0
Netherlands	.3	67.1
Norway	10.9	104.0
United Kingdom	13.0	33.0
Other	.7	7.0
Total	26.4	226.4
EASTERN EUROPE AND U.S.S.R		
U.S.S.R	61.0	1,500.0
Other[1]	2.0	16.0
Total	63.0	1,516.0
MIDDLE EAST		
Bahrain	.2	7.1
Iran	47.9	470.0
Iraq	44.1	29.0
Kuwait[2]	92.5	36.6
Oman	4.0	6.0
Qatar	3.3	148.0
Saudi Arabia[2]	171.5	125.1
United Arab Emirates	33.0	32.8
Other	1.6	1.2
Total	398.1	855.8
AFRICA		
Algeria	8.8	107.1
Cameroon	.6	3.9
Egypt	3.9	7.1
Libya	21.3	21.4
Nigeria	16.6	47.0
Tunisia	1.8	4.2
Other	3.8	7.7
Total	56.8	198.4
FAR EAST AND OCEANIA		
Australia	1.4	18.2
Brunei	1.5	7.4
China	18.4	30.0
India	3.7	16.9
Indonesia	8.5	35.6
Malaysia	3.1	52.7
New Zealand	.2	5.5
Pakistan	.1	15.4
Thailand	.1	5.4
Other	.2	14.4
Total	37.2	201.5
WORLD TOTAL	700.7	3,480.7

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[1] Includes Albania, Bulgaria, Cuba, Czechoslovakia, East Germany, Hungary, Mongolia, North Korea, Poland, Romania, Yugoslavia, and Vietnam.

[2] Includes one-half of the reserves in the Neutral Zone.

Note. Sum of components may not equal total due to independent rounding.

Sources: United States: Energy Information Administration, *U.S. Crude Oil, Natural Gas, and Natural Gas Liquids Reserves, 1985 Annual Report*. All other countries: PennWell Publishing Company, *Oil and Gas Journal*, December 30, 1985, pp. 66-67. The Energy Information Administration does not necessarily certify the data but reproduces it as a matter of convenience.

Table 25. World Estimated Recoverable Reserves of Coal
(Billion Short Tons)

Region Country	Anthracite and Bituminous Coal[1]			Lignite[2]		Total Recoverable Coal
	Recoverable	Portion Surface Minable	Portion of Coking Quality	Recoverable	Portion Surface Minable	
NORTH AMERICA						
Canada	4.18	0.00	1.38	2.33	2.33	6.51
Mexico	1.97	.31	1.37	.00	.00	1.97
United States	248.16	89.28	NA	35.25	35.25	283.41
Other	.00	.00	.00	.00	.00	.00
Total	254.31	89.59	2.75	37.58	37.58	291.90
CENTRAL AND SOUTH AMERICA						
Brazil	14.33	1.43	1.15	.00	.00	14.33
Chile	1.30	NA	NA	.00	.00	1.30
Colombia	1.14	NA	.23	(*)	(*)	1.14
Other	.51	.04	.00	.02	.00	.53
Total	17.28	1.48	1.37	.02	(*)	17.30
WESTERN EUROPE						
Germany, West	32.98	NA	19.78	38.66	38.66	71.64
Greece	.00	.00	.00	1.66	.00	1.66
Turkey	.20	.00	.00	1.90	.00	2.11
United Kingdom	5.06	.00	2.23	.00	.00	5.06
Yugoslavia	1.73	.06	.00	16.50	.00	18.23
Other	3.29	.66	1.03	.58	.07	3.87
Total	43.26	.72	23.04	59.31	38.74	102.58
EASTERN EUROPE AND U.S.S.R						
Bulgaria	.03	NA	.02	4.00	2.65	4.03
Czechoslovakia	3.00	NA	NA	3.15	.00	6.15
Germany, East	.00	.00	.00	NA	NA	NA
Hungary	.23	.00	.00	4.40	.00	4.63
Poland	30.00	.00	6.00	13.20	13.20	43.20
Romania	.00	.00	.00	.00	.00	.00
U.S.S.R.	166.67	34.88	60.00	98.21	97.23	264.88
Total	199.93	34.88	66.02	122.96	113.08	322.89
MIDDLE EAST	.00	.00	.00	.00	.00	.00
AFRICA						
Botswana	3.80	.00	.00	.00	.00	3.80
South Africa	57.03	NA	NA	.00	.00	57.03
Swaziland	2.00	.00	.00	.00	.00	2.00
Other	2.43	.55	.15	.11	(*)	2.54
Total	65.27	.55	.15	.11	(*)	65.38
FAR EAST AND OCEANIA						
Australia	32.52	7.00	13.01	39.90	39.90	72.42
China	108.90	10.91	40.29	.00	.00	108.90
India	NA	NA	NA	1.74	1.65	1.74
Japan	1.10	NA	.57	.02	.00	1.12
Other	1.92	.07	.12	.57	.51	2.49
Total	144.44	17.98	53.99	42.23	42.06	186.67
WORLD TOTAL	724.50	[3] 145.21	[3] 147.33	[3] 262.22	[3] 231.47	986.72

[1] Includes subanthracite and subbituminous.

[2] Includes brown coal.

[3] Sum of reported totals only.

(*) Denotes less than 5 million short tons.

NA Not available.

Note: Sum of components may not equal total due to independent rounding.

Source: The World Energy Conference has compiled these data from questionnaires submitted to National Committees of member countries or corresponding institutions of nonmember countries and from estimates based on examination of existing published sources. The reference year for most reserve data is 1981.

APPENDIX B

WORLD NATURAL GAS & COAL PRODUCTION

Table 29. World Coal Production 1975-1985
(Quadrillion (10^{15}) Btu)

Region Country	1975	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985[1]
NORTH AMERICA											
Canada	0.60	0.59	0.66	0.71	0.78	0.85	0.93	0.98	1.02	1.34	1.41
Mexico	.10	.11	.13	.13	.15	.14	.14	.15	.18	.18	.18
United States	14.99	15.65	15.75	14.91	17.54	18.60	18.38	18.64	17.25	19.72	19.39
Total	15.70	16.36	16.54	15.76	18.46	19.59	19.45	19.77	18.45	21.24	20.98
CENTRAL AND SOUTH AMERICA											
Brazil	.06	.08	.08	.08	.17	.19	.13	.14	.15	.16	.17
Chile	.04	.03	.03	.03	.03	.02	.03	.02	.03	.03	.03
Colombia	.08	.09	.09	.09	.12	.13	.13	.16	.14	.17	.17
Other	.01	.02	.02	.01	.02	.01	.01	.02	.02	.02	.01
Total	.20	.21	.22	.21	.34	.35	.31	.34	.34	.38	.39
WESTERN EUROPE											
Austria	.05	.04	.04	.04	.04	.04	.04	.05	.04	.04	.04
Belgium	.19	.18	.18	.17	.16	.16	.16	.17	.15	.16	.16
France	.68	.67	.65	.63	.56	.54	.55	.51	.50	.48	.43
Germany, West	3.76	3.86	3.70	3.68	3.94	3.96	4.00	4.15	3.94	3.80	3.76
Greece	.09	.12	.12	.12	.12	.12	.14	.14	.16	.17	.18
Italy	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	.01
Norway	.01	.01	.01	.01	.01	.01	.01	.01	.01	.01	.02
Spain	.30	.31	.36	.39	.42	.53	.62	.69	.71	.71	.72
Turkey	.18	.16	.20	.20	.32	.24	.26	.31	.42	.49	.45
United Kingdom	3.04	2.93	2.89	2.92	2.89	3.02	2.96	2.94	2.72	1.17	2.14
Yugoslavia	.42	.44	.46	.47	.49	.55	.61	.64	.68	.74	.74
Other	.01	.01	.01	.01	.01	.01	.01	.02	.01	.01	.01
Total	8.75	8.75	8.65	8.65	8.97	9.20	9.37	9.63	9.35	7.80	8.66
EASTERN EUROPE AND U.S.S.R											
Albania	.01	.01	.01	.01	.02	.02	.02	.02	.02	.02	.03
Bulgaria	.39	.36	.35	.35	.39	.43	.41	.45	.45	.45	.45
Czechoslovakia	1.74	1.78	1.82	1.85	1.86	1.85	1.84	1.87	1.86	1.91	1.90
Germany, East	2.06	2.07	2.11	2.12	2.15	2.17	2.25	2.32	2.36	2.50	2.62
Hungary	.32	.33	.33	.33	.32	.32	.32	.32	.31	.31	.29
Poland	4.52	4.72	4.87	5.03	5.20	5.00	4.27	4.92	5.01	5.10	5.26
Romania	.38	.37	.38	.41	.45	.47	.49	.53	.46	.61	.62
U.S.S.R	13.10	13.30	13.49	13.52	13.44	13.24	13.07	13.31	13.32	13.24	13.47
Total	22.52	22.94	23.38	23.63	23.83	23.50	22.67	23.75	23.82	24.14	24.64
AFRICA											
Morocco	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02
Mozambique	.02	.01	.01	(*)	.01	.01	.01	.02	.01	.01	.01
South Africa	1.64	1.82	2.01	2.13	2.44	2.72	3.08	3.23	3.44	3.82	3.85
Zambia	.02	.02	.02	.02	.01	.01	(*)	.01	.01	.01	.01
Zimbabwe	.06	.07	.07	.07	.08	.08	.07	.07	.08	.06	.05
Other	.01	.01	.02	.03	.03	.01	.02	.02	.02	.02	.02
Total	1.76	1.85	2.15	2.27	2.59	2.85	3.20	3.37	3.58	3.94	3.97
MIDDLE EAST, FAR EAST AND OCEANIA											
Australia	1.91	2.12	2.17	2.22	2.32	2.27	2.60	2.75	2.90	3.06	3.40
China	11.11	11.44	11.82	13.28	13.62	13.33	13.32	14.31	15.36	16.96	18.28
India	1.93	2.04	2.02	2.05	2.09	2.22	2.51	2.62	2.78	2.96	3.02
Indonesia	.01	(*)	.01	.01	.01	.01	.01	.01	.02	.03	.03
Iran	.03	.03	.03	.02	.02	.02	.02	.02	.02	.02	.02
Japan	.59	.46	.46	.48	.44	.45	.44	.42	.41	.40	.39
Korea, North	1.01	1.03	1.03	1.03	1.09	1.14	1.14	1.19	1.14	1.16	1.16
Korea, South	.37	.34	.36	.38	.38	.39	.42	.39	.40	.43	.48
Mongolia	.03	.03	.03	.04	.04	.04	.05	.05	.06	.06	.06
New Zealand	.05	.06	.05	.05	.04	.05	.05	.05	.04	.06	.06
Pakistan	.02	.02	.02	.02	.02	.03	.03	.03	.04	.04	.04
Philippines	(*)	(*)	.01	.01	(*)	.01	.01	.01	.02	.02	.02
Thailand	(*)	.01	(*)	(*)	.01	.01	.02	.01	.02	.02	.05
Vietnam	.11	.14	.15	.15	.15	.16	.16	.16	.17	.14	.14
Other	.09	.09	.08	.08	.08	.07	.07	.07	.06	.06	.05
Total	17.25	17.80	18.24	19.81	20.33	20.20	20.84	22.10	23.43	25.42	27.20
WORLD TOTAL	66.17	68.01	69.17	70.33	74.52	75.68	75.82	78.96	78.97	82.91	85.83

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[1] Preliminary.

(*)Denotes less than one-half the unit of measure.

Note: Coal includes anthracite, subanthracite, bituminous, subbituminous, lignite, and brown coal.

Note: Sum of components may not equal total due to independent rounding.

Sources: Refer to Appendix A.

Table 28. World Dry Natural Gas Production, 1975-1985

(Quadrillion (10^{15}) Btu)

Region Country	1975	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985[1]
NORTH AMERICA											
Canada	2.44	2.46	2.59	2.47	2.66	2.64	2.47	2.44	2.52	2.61	2.83
Mexico	.50	.49	.52	.65	.78	.97	.99	1.07	1.06	1.00	1.01
United States	19.64	19.48	19.57	19.49	20.08	19.90	19.91	18.26	16.53	17.93	16.94
Total	22.59	22.43	22.67	22.61	23.51	23.52	23.16	21.77	20.11	21.54	20.78
CENTRAL AND SOUTH AMERICA											
Argentina	.25	.25	.26	.26	.24	.26	.32	.37	.41	.46	.47
Bolivia	.06	.06	.06	.06	.06	.07	.07	.08	.06	.09	.09
Brazil	.02	.03	.04	.04	.03	.04	.04	.05	.06	.08	.08
Chile	.08	.08	.09	.08	.07	.08	.08	.05	.05	.05	.05
Colombia	.06	.08	.13	.15	.14	.11	.12	.12	.12	.16	.17
Trinidad and Tobago	.05	.07	.08	.10	.13	.08	.08	.11	.13	.20	.21
Venezuela	.48	.46	.44	.46	.52	.56	.59	.68	.66	.69	.70
Other	.02	.02	.02	.15	.25	.17	.05	.05	.05	.05	.05
Total	1.04	1.04	1.13	1.30	1.45	1.39	1.37	1.51	1.55	1.78	1.82
WESTERN EUROPE											
France	.27	.26	.26	.24	.24	.24	.20	.20	.21	.20	.17
Germany, West	.63	.64	.64	.68	.69	.63	.64	.56	.58	.62	.57
Italy	.53	.57	.50	.50	.47	.43	.51	.52	.47	.50	.50
Netherlands	3.05	3.33	2.79	2.38	2.59	3.21	3.00	2.54	2.45	2.52	2.70
Norway	.01	.01	.11	.44	.87	1.01	1.01	1.02	.98	1.06	1.07
United Kingdom	1.34	1.41	1.47	1.38	1.40	1.31	1.30	1.45	1.49	1.51	1.59
Yugoslavia	.06	.07	.07	.08	.07	.07	.07	.07	.06	.08	.08
Other	.10	.09	.13	.08	.12	.08	.06	.05	.06	.06	.06
Total	5.99	6.38	5.97	5.78	6.44	6.99	6.79	6.41	6.30	6.55	6.74
EASTERN EUROPE AND U.S.S.R											
Germany, East	.10	.12	.12	.12	.09	.12	.13	.11	.15	.17	.17
Hungary	.20	.24	.26	.29	.26	.23	.24	.26	.26	.27	.29
Poland	.22	.24	.26	.29	.26	.23	.22	.20	.20	.22	.23
Romania	1.17	1.28	1.35	1.20	1.35	1.35	1.39	1.52	1.57	1.50	1.53
U.S.S.R	9.52	10.56	11.39	12.25	13.38	14.28	15.28	16.44	17.58	19.27	21.09
Other	.05	.05	.05	.05	.04	.03	.04	.04	.04	.04	.04
Total	11.26	12.49	13.42	14.20	15.38	16.24	17.31	18.57	19.80	21.47	23.36
MIDDLE EAST											
Bahrain	.07	.08	.08	.08	.10	.10	.13	.14	.15	.14	.15
Iran	.60	.61	.58	.53	.57	.26	.22	.26	.33	.50	.53
Kuwait	.10	.13	.14	.19	.23	.15	.17	.14	.13	.15	.16
Oman	.00	.00	.00	.00	.00	.00	.08	.10	.03	.05	.05
Qatar	.08	.05	.06	.05	.16	.19	.16	.19	.18	.22	.22
Saudi Arabia	.21	.25	.30	.35	.43	.39	.72	.21	.20	.26	.28
United Arab Emirates	.08	.06	.20	.18	.20	.21	.24	.21	.28	.36	.42
Other	.07	.09	.07	.07	.09	.08	.05	.04	.03	.03	.03
Total	1.22	1.28	1.42	1.45	1.78	1.40	1.78	1.29	1.33	1.72	1.84
AFRICA											
Algeria	.22	.30	.22	.70	.58	.44	.82	1.00	1.39	1.44	1.49
Egypt	(*)	.01	.04	.04	.04	.03	.05	.10	.12	.15	.15
Libya	.12	.14	.14	.19	.19	.19	.12	.13	.15	.17	.16
Nigeria	.01	.02	.02	.02	.05	.04	.08	.05	.09	.10	.10
Other	.10	.03	.02	.02	.03	.03	.03	.04	.04	.05	.05
Total	.46	.50	.44	.97	.89	.73	1.10	1.31	1.79	1.91	1.96
FAR EAST AND OCEANIA											
Australia	.19	.22	.25	.27	.29	.33	.39	.40	.40	.42	.43
Bangladesh	.02	.03	.03	.04	.05	.05	.06	.06	.07	.07	.07
Brunei	.21	.28	.32	.33	.33	.35	.38	.36	.37	.34	.33
China	.34	.38	.43	.53	.54	.53	.47	.40	.45	.46	.50
India	.03	.04	.05	.06	.04	.05	.07	.09	.11	.12	.12
Indonesia	.09	.13	.21	.21	.41	.66	.69	.70	.81	1.11	1.26
Japan	.09	.09	.11	.10	.04	.04	.04	.07	.08	.09	.10
Malaysia	.02	.02	.03	.04	.01	.04	.04	.06	.15	.34	.36
New Zealand	.01	.04	.06	.05	.05	.05	.05	.10	.11	.11	.14
Pakistan	.16	.16	.17	.18	.23	.28	.31	.34	.33	.34	.34
Thailand	.00	.00	.00	.00	.00	.00	.00	.05	.06	.07	.07
Other	.17	.17	.16	.12	.13	.12	.15	.15	.16	.17	.08
Total	1.34	1.56	1.82	1.93	2.11	2.49	2.65	2.79	3.10	3.63	3.79
WORLD TOTAL	43.90	45.68	46.88	48.24	51.57	52.75	54.16	53.66	53.97	58.61	60.28

[1] Preliminary

(*)Denotes less than one-half the unit of measure

Note: Sum of components may not equal total due to independent rounding

Sources: Refer to Appendix A

Table 21. World Coal Production, 1984
(Thousand Short Tons)

Region Country	Primary[1]			Secondary[1]		
	Anthracite	Bituminous Coal	Lignite	Metallurgical Coke	Anthracite & Bituminous Briquets	Lignite Briquets
NORTH AMERICA						
Canada	—	52,342	10,932	5,342	—	—
Mexico	—	9,920	—	2,535	—	—
United States	4,162	828,689	63,070	30,561	—	—
Total	4,162	890,951	74,002	38,438	—	—
CENTRAL AND SOUTH AMERICA						
Brazil	—	7,885	—	6,959	—	—
Chile	—	1,305	45	330	—	—
Colombia	—	[2] 7,564	—	330	—	—
Other	140	617	—	524	—	—
Total	140	17,371	45	8,143	—	—
WESTERN EUROPE						
Austria	—	—	3,228	2,043	—	—
Belgium	112	6,830	—	6,532	48	—
France	2,597	15,695	2,674	9,920	1,596	—
Germany, West	6,390	87,160	139,706	23,663	1,584	7,024
Greece	—	—	34,806	17	—	203
Italy	—	—	1,929	7,863	—	—
Norway	—	497	—	347	—	—
Spain	6,100	10,742	26,814	3,618	18	—
Turkey	248	7,829	29,580	2,760	—	25
United Kingdom	1,320	53,245	—	7,696	188	—
Yugoslavia	—	429	71,302	3,875	—	1,045
Other	210	92	—	4,682	—	440
Total	16,977	182,519	310,039	73,016	3,434	8,737
EASTERN EUROPE AND U.S.S.R						
Albania	—	—	1,956	18	—	—
Bulgaria	93	153	35,420	1,307	—	1,335
Czechoslovakia	—	29,124	113,380	11,355	—	1,178
Germany, East	—	—	326,660	1,300	—	61,795
Hungary	—	2,836	24,773	775	—	1,710
Poland	—	[2] 211,194	55,532	18,866	627	165
Romania	—	[2] 9,323	39,485	4,960	825	1,040
U.S.S.R	—	[2] 613,436	171,740	94,800	225	6,095
Total	93	866,066	768,946	133,381	1,677	73,318
MIDDLE EAST						
Iran	—	1,000	—	385	—	—
Total	—	1,000	—	385	—	—
AFRICA						
Morocco	920	—	—	—	—	—
South Africa	4,260	174,352	—	2,040	—	—
Zimbabwe	—	2,535	—	254	—	—
Other	138	1,780	—	1,050	—	—
Total	5,318	178,667	—	3,344	—	—
FAR EAST AND OCEANIA						
Australia	—	[2] 114,866	38,028	3,927	—	902
China	170,000	666,785	33,190	39,848	—	—
India	—	159,692	8,432	12,091	—	159
Indonesia	—	1,196	—	—	—	—
Japan	25	17,040	1,282	56,520	337	—
Korea, North	40,000	10,000	500	3,750	19,672	—
Korea, South	22,750	—	—	5,730	—	—
Mongolia	—	450	6,000	—	—	—
New Zealand	—	2,525	259	18	6	—
Pakistan	—	2,060	—	—	—	—
Philippines	—	1,257	—	—	—	—
Thailand	—	—	2,576	—	—	—
Vietnam	5,500	—	—	—	—	—
Other	—	2,430	38,274	—	40	—
Total	238,275	978,301	128,541	121,884	20,055	1,061
WORLD TOTAL	264,965	3,114,875	1,281,573	378,591	25,166	83,116

[1] Primary coal includes all coal mined and when necessary washed and sorted. Secondary coal (e.g. coke, briquets), is derived from primary coal.

[2] Includes anthracite.

-- Indicates data not applicable.

Note: Sum of components may not equal total due to independent rounding.

Sources: Refer to Appendix A.

APPENDIX C

ENERGY PRICES IN SELECTED OECD COUNTRIES

Table 17. Energy Prices (Including Taxes) In Selected OECD Countries, 1978-1985

Region Country	Automotive Fuels		Light Fuel Oil		Heavy Fuel Oil		Natural Gas			Electricity		Coal	Steam Coal		
	Premium Leaded Gasoline	Diesel Fuel	Residential	Industrial	Industrial	Electricity Generation	Industrial	Electricity Generation	Residential	Residential	Industrial	Metallurgical	Residential	Industrial	Electricity Generation
	U.S. Dollars per Gallon				U.S. Dollars per Barrel		U.S. Dollars per 1,000 Cubic Feet			U.S. Cents per Kilowatthour		U.S. Dollars per Short Ton			
NORTH AMERICA															
United States															
1978	[1] 0.66	0.54	0.48	0.39	[2] 11.79	[2] 13.28	1.54	1.45	2.58	[2] 4.30	[2] 2.76	44.97	NA	30.45	23.06
1979	[1] .87	73	.66	.55	[2] 15.98	[2] 18.02	2.03	1.73	3.22	[2] 4.63	[2] 3.01	53.86	NA	32.31	25.90
1980	[1] 1.23	1.01	.94	.82	[2] 21.05	[2] 26.07	2.48	2.22	3.93	[2] 5.29	[2] 3.64	56.23	NA	35.13	28.75
1981	[1] 1.37	1.18	1.17	1.06	[2] 28.51	[2] 33.15	2.97	2.84	4.22	[2] 6.14	[2] 4.22	62.45	NA	39.51	32.07
1982	[1] 1.32	1.16	1.15	1.04	[2] 25.85	[2] 30.45	3.70	3.41	5.11	[2] 6.83	[2] 4.93	64.77	NA	41.00	34.90
1983	[1] 1.25	1.20	[2] 1.05	[2] .88	[2] 25.71	[2] 28.78	4.25	3.55	6.19	[2] 7.14	[2] 5.02	59.54	NA	38.88	35.00
1984	[1] 1.22	1.22	[2] 1.06	[2] .86	[2] 27.68	[2] 30.25	4.14	3.64	6.38	[2] 7.50	[2] 5.02	56.81	NA	39.25	35.11
1985	[1] 1.20	1.17	[2] 1.03	[2] .83	[2] 24.86	[2] 25.60	3.99	3.64	6.42	[2] 7.82	[2] 5.19	54.31	NA	37.24	34.55
1Q85	[1] 1.15	1.18	[2] 1.05	[2] .82	[2] 27.93	[2] 28.68	4.19	3.79	5.98	[2] 7.28	[2] 5.12	55.15	NA	38.29	34.68
2Q85	[1] 1.20	1.16	[2] 1.05	[2] .85	[2] 26.00	[2] 26.98	3.96	3.76	6.11	[2] 7.73	[2] 5.09	54.48	NA	37.73	35.05
3Q85	[1] 1.24	1.14	[2] .98	[2] .79	[2] 22.93	[2] 23.36	3.83	3.60	7.07	[2] 8.24	[2] 5.36	53.69	NA	36.29	34.29
4Q85	[1] 1.20	1.19	[2] 1.03	[2] .86	[2] 22.60	[2] 23.38	3.99	3.41	6.51	[2] 8.05	[2] 5.19	53.91	NA	36.65	34.20
Canada															
1978	[1] .69	NA	.41	.41	10.69	10.53	1.44	1.14	2.51	2.5	1.7	57.20	NA	33.36	29.93
1979	[1] .73	NA	.45	.45	12.40	11.34	1.52	1.08	2.39	2.6	1.9	58.45	NA	39.11	35.60
1980	[1] .85	NA	.54	.55	14.33	11.99	1.69	1.09	2.68	2.9	2.1	60.19	NA	39.49	38.40
1981	[1] 1.13	1.27	.74	.75	19.08	17.10	2.30	1.53	3.34	3.1	1.9	62.77	NA	43.89	41.07
1982	[1] 1.37	1.27	.89	.90	20.97	19.24	2.68	1.99	3.80	3.3	2.2	64.32	NA	46.98	44.47
1983	[1] 1.47	1.33	.99	1.00	24.47	23.22	2.96	NA	4.46	3.7	2.3	64.82	NA	49.11	NA
1984	[1] 1.48	1.30	1.01	.82	24.95	NA	2.80	NA	4.31	NA	NA	54.61	NA	48.25	NA
1985	[1] 1.45	1.31	1.05	.71	24.54	NA	2.68	NA	4.16	NA	NA	50.16	NA	46.28	NA
1Q85	[1] 1.52	1.32	1.06	.73	26.16	NA	2.83	NA	3.90	NA	NA	51.53	NA	47.42	NA
2Q85	[1] 1.46	1.30	1.05	.73	25.35	NA	2.68	NA	4.04	NA	NA	49.95	NA	45.61	NA
3Q85	[1] 1.43	1.29	1.05	.70	24.42	NA	2.59	NA	4.83	NA	NA	49.96	NA	46.46	NA
4Q85	[1] 1.41	1.33	1.04	.69	22.22	NA	2.62	NA	3.88	NA	NA	49.22	NA	45.62	NA
WESTERN EUROPE															
France															
1978	2.15	1.30	.72	.68	13.98	NA	2.35	NA	6.88	8.1	3.5	92.63	146.82	54.93	NA
1979	2.56	1.65	.93	.89	18.33	NA	2.73	NA	7.62	9.4	4.1	105.68	183.93	69.42	NA
1980	3.03	2.11	1.39	1.33	27.39	NA	4.19	NA	9.86	11.5	5.2	119.41	261.40	89.01	47.12
1981	2.70	1.98	1.40	1.34	30.06	NA	4.45	NA	9.68	9.8	4.4	109.54	253.84	84.35	54.31
1982	2.56	1.88	1.37	1.31	28.58	NA	4.48	NA	9.74	9.5	4.2	100.40	240.07	75.24	50.17
1983	2.37	1.76	1.30	1.25	27.49	NA	4.22	NA	9.17	9.1	4.1	96.00	228.68	72.81	43.40
1984	2.24	1.63	1.19	1.14	29.73	NA	3.97	NA	8.50	8.7	3.8	91.57	225.41	68.39	38.37
1985	2.39	1.69	1.29	1.24	27.25	NA	NA	NA	NA	NA	NA	NA	240.27	NA	NA
1Q85	2.12	1.52	1.14	1.10	28.39	NA	NA	NA	NA	NA	NA	NA	210.67	NA	35.77
2Q85	2.31	1.69	1.33	1.28	28.59	NA	NA	NA	NA	NA	NA	NA	224.85	NA	NA
3Q85	2.52	1.70	1.30	1.24	25.44	NA	NA	NA	NA	NA	NA	NA	243.44	NA	NA
4Q85	2.63	1.86	1.41	1.35	26.58	NA	NA	NA	NA	NA	NA	NA	282.10	NA	NA
Germany, West															
1978	1.75	1.48	.58	.52	15.59	15.59	2.55	1.86	5.92	8.5	4.7	56.69	183.39	82.79	75.39
1979	2.06	1.77	1.04	.92	20.62	21.95	2.88	2.20	6.58	9.5	5.3	59.44	231.72	93.66	85.77
1980	2.43	2.10	1.30	1.15	28.01	29.62	3.74	3.02	7.99	10.0	5.8	64.69	262.47	109.01	89.63
1981	2.38	1.90	1.20	1.06	32.19	32.83	3.89	3.42	7.75	9.1	5.2	72.67	236.76	102.74	87.44
1982	2.17	1.81	1.23	1.09	28.48	28.75	4.93	3.77	8.47	9.2	5.3	72.69	233.99	101.26	86.34
1983	2.04	1.66	1.07	.94	26.88	27.19	4.49	3.46	7.97	9.1	5.1	62.05	227.79	99.32	81.19
1984	1.87	1.53	1.02	.89	28.81	28.95	4.31	3.49	7.18	8.5	4.7	59.80	209.91	92.03	74.38
1985	1.84	1.51	1.02	.89	27.12	27.38	NA	NA	NA	8.4	4.7	57.54	210.30	89.74	74.35
1Q85	1.62	1.36	1.01	.89	29.47	30.96	NA	NA	NA	7.4	4.2	51.62	188.33	80.51	67.81
2Q85	1.74	1.49	.98	.86	28.04	26.46	NA	NA	NA	7.8	4.4	54.46	198.68	84.94	70.95
3Q85	1.93	1.49	.97	.85	24.23	24.25	NA	NA	NA	8.7	4.9	59.00	213.34	92.02	75.59
4Q85	2.08	1.71	1.12	.98	26.75	25.86	NA	NA	NA	9.6	5.4	65.06	240.85	101.47	83.07

See footnotes at end of table

Table 17. Energy Prices (Including Taxes) In Selected OECD Countries, 1978-1985 (Continued)

Region Country	Automotive Fuels		Light Fuel Oil		Heavy Fuel Oil		Natural Gas			Electricity		Coal	Steam Coal		
	Premium Leaded Gasoline	Diesel Fuel	Residential	Industrial	Industrial	Electricity Generation	Industrial	Electricity Generation	Residential	Residential	Industrial	Metallurgical	Residential	Industrial	Electricity Generation
	U.S. Dollars per Gallon				U.S. Dollars per Barrel		U.S. Dollars per 1,000 Cubic Feet			U.S. Cents per Kilowatthour		U.S. Dollars per Short Ton			
WESTERN EUROPE															
Italy															
1978	2.23	0.64	0.66	0.59	13.06	12.86	1.98	1.95	4.36	4.6	1.9	56.37	151.77	33.99	33.18
1979	2.47	.80	.85	.75	17.94	17.96	2.80	2.80	5.46	5.2	2.3	58.03	154.79	40.76	34.74
1980	3.10	1.21	1.37	1.20	27.27	27.08	4.13	4.19	5.01	7.1	3.1	62.18	188.39	57.12	46.07
1981	2.97	1.11	1.24	1.08	30.11	31.03	4.69	4.68	4.98	6.9	2.6	69.14	194.65	65.08	63.91
1982	2.88	1.19	1.33	1.18	26.67	27.27	4.19	4.03	6.90	7.1	2.5	71.54	164.65	58.56	63.25
1983	2.93	1.26	1.40	1.21	26.68	26.44	4.23	4.01	8.23	7.6	2.7	61.71	153.14	43.25	53.04
1984	2.79	1.20	1.33	1.16	28.80	28.16	4.31	4.15	7.50	1.1	5.7	NA	NA	NA	42.74
1985	2.63	1.21	1.33	1.12	26.13	26.05	NA	NA	NA	NA	6.0	NA	NA	NA	NA
1Q85	2.43	1.07	1.20	1.02	27.33	27.49	NA	4.31	NA	NA	5.8	NA	NA	NA	41.49
2Q85	2.55	1.23	1.39	1.18	29.37	28.58	NA	3.53	NA	NA	5.6	NA	NA	NA	47.18
3Q85	2.69	1.19	1.27	1.07	22.95	24.69	NA	3.06	NA	NA	6.0	NA	NA	NA	47.14
4Q85	2.85	1.35	1.45	1.23	24.88	23.45	NA	NA	NA	NA	6.5	NA	NA	NA	NA
United Kingdom															
1978	1.22	1.24	.62	.48	15.35	13.78	2.13	1.62	3.43	5.3	3.8	NA	75.84	40.48	39.14
1979	1.79	1.70	.86	.72	21.29	19.04	2.59	2.15	3.95	6.3	4.6	NA	97.95	53.78	50.59
1980	2.50	2.19	1.29	1.13	32.08	30.13	3.87	2.85	5.25	8.8	6.3	NA	136.94	74.35	68.23
1981	2.57	2.23	1.31	1.12	33.58	31.28	4.02	5.29	5.54	9.1	6.3	NA	137.84	72.79	68.65
1982	2.42	2.05	1.33	1.06	30.88	28.33	3.78	4.12	5.89	8.6	6.0	NA	126.70	75.98	63.68
1983	2.25	1.87	1.20	.95	29.05	27.16	3.32	3.61	5.68	7.7	5.2	NA	115.61	68.20	59.07
1984	2.05	1.68	1.02	.86	30.80	NA	3.01	3.19	5.17	6.9	4.6	NA	109.75	60.22	NA
1985	2.13	1.78	1.06	.89	30.50	NA	3.15	NA	5.23	6.8	NA	NA	111.65	60.04	NA
1Q85	1.79	1.52	.95	.83	31.38	NA	2.66	NA	4.02	5.6	4.2	NA	97.63	50.70	NA
2Q85	2.13	1.79	1.11	.90	32.86	NA	3.02	NA	5.06	6.4	4.2	NA	104.73	57.69	NA
3Q85	2.28	1.88	1.06	.88	29.25	NA	3.30	NA	6.28	7.6	4.6	NA	115.92	63.18	NA
4Q85	2.31	1.95	1.11	.97	28.50	NA	3.62	NA	5.54	7.7	NA	NA	128.32	68.58	NA
Sweden															
1978	1.56	.62	.53	.53	16.62	13.83	NC	NC	NC	4.7	2.9	53.58	109.70	64.77	43.20
1979	1.94	.83	.76	.79	25.02	22.97	NC	NC	NC	5.1	3.5	58.06	131.82	72.50	57.91
1980	2.64	1.31	1.22	1.20	35.27	29.87	NC	NC	NC	5.9	4.0	60.77	148.50	77.66	79.30
1981	2.64	1.35	1.25	1.27	39.88	32.87	NC	NC	NC	5.4	3.4	70.73	141.97	78.94	78.37
1982	2.40	1.41	1.30	1.33	37.82	27.81	NC	NC	NC	4.5	2.9	71.82	135.04	72.39	70.53
1983	2.07	1.26	1.14	1.14	36.61	27.02	NC	NC	NC	3.9	2.5	61.72	122.55	48.35	48.56
1984	1.93	1.31	1.15	1.13	40.68	28.58	NC	NC	NC	3.9	3.1	56.77	128.27	50.35	42.99
1985	2.06	1.34	1.18	1.16	40.74	24.92	NC	NC	NC	NA	NA	56.79	138.89	58.19	43.91
1Q85	1.83	1.24	1.08	1.07	40.21	29.19	NC	NC	NC	NA	NA	55.22	126.21	56.60	37.11
2Q85	2.00	1.38	1.20	1.19	43.02	23.89	NC	NC	NC	NA	NA	55.70	130.77	56.31	39.87
3Q85	2.21	1.40	1.16	1.12	39.55	22.79	NC	NC	NC	NA	NA	58.79	139.30	56.95	47.12
4Q85	2.19	1.36	1.28	1.26	40.18	23.81	NC	NC	NC	NA	NA	57.47	159.28	62.92	51.55
Netherlands															
1978	1.89	.92	.67	.54	14.89	13.70	2.09	1.88	3.84	8.2	4.1	93.15	NA	52.50	37.22
1979	2.25	1.20	.93	.75	17.84	19.64	2.60	2.60	4.41	9.6	5.0	104.98	NA	55.89	39.92
1980	2.73	1.56	1.35	1.13	28.04	28.91	3.48	3.88	5.31	11.5	5.9	116.27	NA	57.12	58.10
1981	2.57	1.45	1.40	1.07	33.05	32.82	3.95	4.15	5.55	10.9	5.8	106.98	NA	64.09	73.74
1982	2.47	1.43	1.34	1.10	29.77	29.02	4.11	4.19	6.07	10.5	5.6	110.08	NA	65.31	77.36
1983	2.30	1.27	1.15	.98	28.11	27.57	3.79	3.86	6.05	9.7	5.1	92.49	NA	57.14	64.41
1984	2.14	1.20	1.12	NA	29.98	30.21	NA	NA	5.74	8.8	NA	NA	NA	NA	NA
1985	2.11	1.19	1.12	NA	27.49	NA	NA	NA	5.88	NA	NA	NA	NA	NA	NA
1Q85	1.83	1.09	1.03	NA	29.93	NA	NA	NA	5.26	NA	NA	NA	NA	NA	NA
2Q85	2.08	1.17	1.11	NA	28.97	NA	NA	NA	5.56	NA	NA	NA	NA	NA	NA
3Q85	2.21	1.16	1.07	NA	24.88	NA	NA	NA	6.05	NA	NA	NA	NA	NA	NA
4Q85	2.32	1.34	1.26	NA	26.18	NA	NA	NA	6.66	NA	NA	NA	NA	NA	NA

See footnotes at end of table

Table 17. Energy Prices (Including Taxes) In Selected OECD Countries, 1978-1985 (Continued)

Region Country	Automotive Fuels		Light Fuel Oil		Heavy Fuel Oil		Natural Gas			Electricity		Coal	Steam Coal		
	Premium Leaded Gasoline	Diesel Fuel	Residential	Industrial		Electricity Generation	Industrial	Electricity Generation	Residential		Industrial	Metallurgical	Residential	Industrial	Electricity Generation
	U.S. Dollars per Gallon				U.S. Dollars per Barrel		U.S. Dollars per 1,000 Cubic Feet			U.S. Cents per Kilowatthour		U.S. Dollars per Short Ton			
FAR EAST															
Australia															
1978	0.80	0.75	0.58	0.50	13.79	13.79	0.97	NA	3.67	3.9	2.7	26.16	NA	NA	9.25
1979	1.06	.91	.76	.65	17.09	17.09	1.10	NA	3.52	4.1	2.8	26.47	NA	NA	10.53
1980	1.32	1.26	1.14	1.00	29.89	29.89	.83	NA	4.13	4.6	3.2	26.07	NA	NA	12.37
1981	1.55	1.46	1.34	1.18	36.62	36.62	.99	NA	4.56	5.2	3.7	29.34	NA	NA	15.33
1982	1.49	1.44	1.41	1.20	36.19	36.19	1.13	NA	4.43	5.7	4.1	33.06	NA	NA	18.99
1983	1.49	1.53	1.46	1.39	35.91	35.91	1.31	NA	4.37	5.7	3.8	28.50	NA	28.46	18.40
1984	1.58	1.55	NA	NA	NA	NA	1.47	NA	4.65	5.8	3.8	25.36	NA	27.86	19.80
1985	1.39	1.29	NA	NA	NA	NA	2.12	NA	4.07	5.0	3.2	NA	NA	NA	NA
1Q85	1.39	1.35	NA	NA	NA	NA	2.21	NA	4.29	5.3	3.3	20.92	NA	27.25	18.05
2Q85	1.35	1.25	NA	NA	NA	NA	1.99	NA	3.84	4.7	2.9	20.93	NA	21.48	16.37
3Q85	1.45	1.31	NA	NA	NA	NA	2.15	NA	4.04	5.0	3.2	23.27	NA	22.48	15.99
4Q85	1.38	1.24	NA	NA	NA	NA	2.12	NA	4.11	4.9	3.1	NA	NA	NA	NA
Japan															
1978	[1] 2.00	NA	.81	.60	14.12	16.58	9.33	2.26	11.74	9.4	6.3	54.33	NA	34.72	45.39
1979	[1] 2.10	NA	.90	.65	17.44	25.00	8.94	3.09	11.40	9.3	6.2	54.94	NA	37.57	44.84
1980	[1] 2.58	NA	1.49	1.15	37.47	37.69	10.70	4.66	14.72	11.8	8.7	60.07	NA	47.26	50.02
1981	[1] 2.64	NA	1.62	1.32	40.05	44.16	11.87	5.67	16.45	13.2	10.0	64.45	NA	58.94	58.62
1982	[1] 2.60	NA	1.59	1.36	36.80	40.17	10.54	5.39	14.61	11.9	9.1	67.56	NA	56.94	55.88
1983	[1] 2.55	NA	1.63	1.38	34.00	36.96	10.79	4.94	15.21	12.5	9.5	61.61	NA	50.44	56.77
1984	[1] 2.39	NA	1.50	1.33	31.57	35.41	10.38	4.82	15.19	12.6	9.5	57.23	NA	45.16	56.43
1985	[1] 2.34	NA	1.41	1.28	32.69	NA	10.24	NA	15.26	12.6	9.0	54.28	NA	41.56	NA
1Q85	[1] 2.19	NA	1.32	1.22	30.67	NA	9.41	NA	14.03	11.6	8.3	54.92	NA	41.72	NA
2Q85	[1] 2.23	NA	1.33	1.22	32.49	NA	9.67	NA	14.42	11.9	8.5	53.80	NA	41.57	NA
3Q85	[1] 2.30	NA	1.40	1.28	32.52	NA	10.16	NA	15.15	12.5	8.9	53.56	NA	40.94	NA
4Q85	[1] 2.65	NA	1.60	1.41	35.07	NA	11.71	NA	17.46	14.4	10.3	54.85	NA	42.01	NA

[1] Unleaded regular gasoline

[2] Excludes taxes

NA Not available.

NC No appreciable consumption at the national level.

Note Prices are representative prices for each country based on quarterly data averaged for the year. Included are non-refundable taxes and custom duties paid by consumers after rebates, subsidies, etc.

Sources Refer to Appendix A

PROJECT REPORT

HATCH ASSOCIATES LTD. CONSULTING ENGINEERS
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STELCO - LB PROCESS EVALUATION

December 8, 1987

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REVIEW OF DRI/LIQUID IRONMAKING PROCESSES

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S. Kumar
S. Kumar

SUMMARY & CONCLUSIONS

This report is a review of the major direct reduction and liquid ironmaking processes (both existing and emerging) with which the LB process would have to compete. The objective is to determine the commercial potential for the LB process for iron.

Because of facility sizing considerations, the LB will not be a competitor to the blast furnace for high tonnage production of iron.

Although numerous directly reduced iron (DRI) processes have been developed, the Midrex and HYL processes, both gas-based, account for about 90% of world DRI production. The SL/RN & ACCAR processes (kiln reactors) are the major coal based processes. There are no fully commercialized processes using ore fines and coal raw material feeds, as the LB would. The use of coal and ore fines eliminates the need for coke ovens and agglomeration plants. Non-coking coal is available in many countries where coking coal (required for blast furnaces) and natural gas (required for the gas-based DRI processes) are not available.

There are no commercialized DRI processes which use an indirect heating chamber as in the LB. Most processes use a shaft, kiln or fluidized bed type reactor. Processes similar to the LB, in that they also require coal and ore fines, are the New Zealand Steel operation (based on SL/RN kiln and ironsands) and Elred process. The LB process does not offer any advantages over these two processes in terms of the type of ore and energy.

At this time the competitiveness of the LB process with respect to capital cost, operating cost and process operability is unknown. The net specific energy consumption, another key parameter for the LB process, would have to be in the range of 3.5 - 5 Gcal/tonne liquid steel in order to be competitive. If the major LB application is in developing countries, the equipment must not be overly complex, so that high operating factors may be achieved.

1.0 INTRODUCTION

The purpose of this review of ironmaking processes, with the focus on the direct reduction (DR) processes, is to identify the potential for the LB process for the production of virgin iron. The LB process for iron production is based on coal and ore fines as feed. The complete process envisages the direct reduction to sponge which is then charged, without loss of its sensible heat, directly to an electric furnace for melting and final reduction.

The proposed LB process can be considered as one of the multitude of direct reduction processes which can produce either DRI or liquid iron. The DR processes compete with both the blast furnace production and the steel produced from scrap from the electric arc furnaces (EAF). The LB process has been considered to have potential use for urban waste disposal, coke production, and for the recycling/reclamation of dust from EAF operations. None of these will be considered in this review, although EAF dust processing will be discussed separately.

DRI has a history of almost a century as a serious possibility. The first industrial patent is believed to be the 1896 invention by Henry Jones of Brooklyn. There appears to have been a renewed interest in about 1920, after which over 100 DRI processes have been developed. Commercial significance, however, really only came into being in the late 1950's. Midrex Corporation has compiled a list (see Appendix A) of steelmaking grade DRI plants built, under construction or planned. This list gives a total of 77 plants since 1957, which are fueled as follows: 53 gas; 2 gas plus oil or coal, 20 coal, one oil and one kerosene.

In the 25 years to 1982, the list has 50 plants which were built and 29 of these are still operating. Gas installations accounted for about 75%, and 60% of these are still operating, as are 60% of the coal fueled plants. Since 1983, more of the plants built, under construction or planned, are fueled by coal, i.e. 10 out of 27 installations. Even more significantly, of the plants into production since 1982, 8 out of 15 are coal fueled. This may not be a sustainable trend but it perhaps does give some encouragement for the coal fueled LB process.

The approach taken in this study has been to assess the basic features (ore and coal feed, energy requirements, process operability, capital and operating costs, state of development, etc.) of a process based upon data from published literature and in-house information. Detailed descriptions of the processes are not presented here as they have been well-covered in the literature. References 1-4 provide process flowsheets and descriptions.

In this report, DRI refers to sponge iron and all tonnages are metric tonnes.

2.0 WHY AN ALTERNATIVE IRONMAKING PROCESS

The Blast Furnace has been used as the primary process for the production of liquid iron for several centuries. Its inherent advantages include:

- good overall thermal efficiency
- high reducing ability
- high productivity
- long life

However, its major weaknesses are:

- the need for high quality ore and coke (which may be unavailable or expensive)
- the need for ore agglomeration facilities and coke ovens (which require significant investment)
- it is generally uneconomical for low annual production rates (below approximately 1 million tpy).
- environmental problems associated with both the blast furnace and coke ovens

Low grade ores need to be beneficiated for use in the production of iron. This typically consists of crushing fines, screening and then concentrating (separation of iron oxide from gangue) the ore. The concentrates (fines) must be agglomerated to pellets or sinter for use in blast furnaces or most DRI processes.

A typical cumulative size analysis of iron ore concentrate from a Canadian source is:

SIZE		CUMULATIVE WEIGHT
MESH	MICRONS	
+ 14	+ 1168	1 %
+ 20	+ 833	3
+ 28	+ 589	10
+ 48	+ 295	38
+ 65	+ 208	60
+100	+ 147	78
+150	+ 104	92
+200	+ 74	98
+400	+ 37	99.5

Modern practice is to make pellets in the 12.5 mm range, with a normal size requirement at the point of receipt of 90% - 16 mm + 9.5 mm.

Typical size analysis (Reference 10) of the Wabush ore concentrates used in experiments for the LB process is:

MESH SIZE	CUMULATIVE WEIGHT
+ 20	0.0 %
+ 30	9.3 %
+ 50	35.4 %
+100	64.8 %

The availability of metallurgical coal is limited in many regions of the world. An alternative in regions lacking an economic coke supply is gas or non-coking coal based Direct Reduction processes. In many important steelmaking regions, non-coking coal is more available and cheaper than natural gas (see Reference 12). If steel scrap is economically available, scrap melting technologies can be used for the production of steel.

A process, such as the LB, which uses concentrates directly, eliminates the capital and operating costs of an agglomeration plant. However, drying of the concentrates would still be required. The direct use of coal eliminates the capital and operating costs of coke ovens, or the need for cheap natural gas which would be required with a gas based DRI process.

The capital costs which can be avoided by the LB process is illustrated by considering the costs estimated in a recent review (Reference 4) for coal-based ironmaking where coking and sintering are necessary. Two levels of annual production are given to show the range of capital costs per annual tonnage of production. However, a process based on coal fines will have a cost for grinding the coal.

Capital Cost per Annual Tonne of Hot Metal Production

Production Capacity (million tonnes/year)	0.5	3.0
Coke Ovens	US$ 220/annual tonne	US$ 140/annual tonne
Sinter Plant	100	50
Sub Total:	320	190
Blast Furnace	300	140
TOTAL:	US$ 620/annual tonne	US$ 330/annual tonne

The larger capacity plants are usually dictated by the economic plant size required for a hot mill complex and particularly where the production includes flat rolled products for automotive and appliance usage. A size of 2 to 3 million tpy is required for economical production with current technology. There is no existing competitor to the blast furnace for iron production of over 1 million tpy in a single unit. The alternative for iron making for lower production levels have been direct reduction processes.

There is a need for an ironmaking process which uses fine ore feed and coal. The LB process, in principle, meets these needs.

3.0 WHAT IS DIRECT REDUCTION

Direct Reduction is a process for metallic iron production by reduction of iron oxide below the melting temperature of any materials involved. The product of most DR processes is sponge iron (solid form), which is then charged to a separate melting furnace. The melting of sponge iron requires a higher specific energy consumption than the use of 100% scrap. This is because of the additional energy requirements for reducing the remaining iron oxides and the presence of gangue which must be compensated for by additional lime bearing units to produce a basic steelmaking slag.

As a general rule it is preferable to magnetically separate the sponge whenever the energy cost to a) melt and reduce the iron oxide, and b) to heat the lime, and c) to dissolve the gangue in a basic slag is more than the cost of the heat lost by cooling hot sponge prior to separation. Operating experience indicates that even at a low fraction (10%) of unreacted iron oxide and gangue material, the power to melt starts to increase rapidly above that predicted on a theoretical basis.

Many processes such as Corex and Combismelt, produce liquid iron (i.e. these processes incorporate a reduction and a melting unit). Reference 1 gives a detailed description of the metallurgy of direct reduction.

4.0 USES FOR DRI

Potential major uses for DRI include:

a) Steelmaking in electric arc furnaces. This is by far the largest use.

b) Steelmaking in BOPs. This is not being done on a significant scale.

c) Iron melting in cupolas.

d) Ironmaking in blast furnaces. The use of DRI reduces coke rate and increases hot metal production rate.

An advantage of DRI over scrap steel is that it is relatively free from tramp elements such as copper, zinc, tin, chromium and others that are usually present in scrap. The absence of these tramp elements is very important for high grade steels, tinplate and steel which will undergo drawing. DRI can be used as a diluent with scrap to produce lower residual liquid for steel and/or foundry products.

5.0 HISTORICAL PRODUCTION & CAPACITY

In comparison with blast furnace ironmaking, direct reduction plants are relatively new. The earliest plants, using the Wiberg and Hoganas processes, started up in Sweden in the early 1950's. Since then, about 100 different processes (at various stages of commercialization) have been developed. However, only a few represent a significant portion of current world production.

The earliest installation of the Midrex process, (a gas-fueled process) currently the dominant DRI process, was at Gilmore Steel (Portland, Oregon) in 1969. From 1969 there have been 27 Midrex installations built or planned with 19 now in production. The earliest installation of a HYL process was at Hylsa, Monterrey in 1957. Since 1957 there have been 19 HYL installations built or planned with 11 currently in production. The only other operating gas-fueled plant is the Fior plant in Venezuela.

Figure 1 gives the world installed DRI capacity (steelmaking grade) and production. The 1986 capacity was 23.6 million tpy and production was 12.7 million tpy, for a capacity utilization of 54%. The 1986 world steel production was about 715 million tpy; therefore, DRI represented a relatively small component of total production. Further data is given in Appendix A.

Figure 2 gives a breakdown by process of DRI production. The Midrex and HYL processes (gas-based) dominate, accounting for nearly 90% of DRI production in 1984. The coal based processes accounted for 8% of 1984 production.

Figure 3 illustrates 1986 DRI production by country/region. The largest producer was Venezuela, followed by Mexico, Indonesia, Saudi Arabia, Argentina, USSR and Canada.

DRI PRODUCTION AND CAPACITY

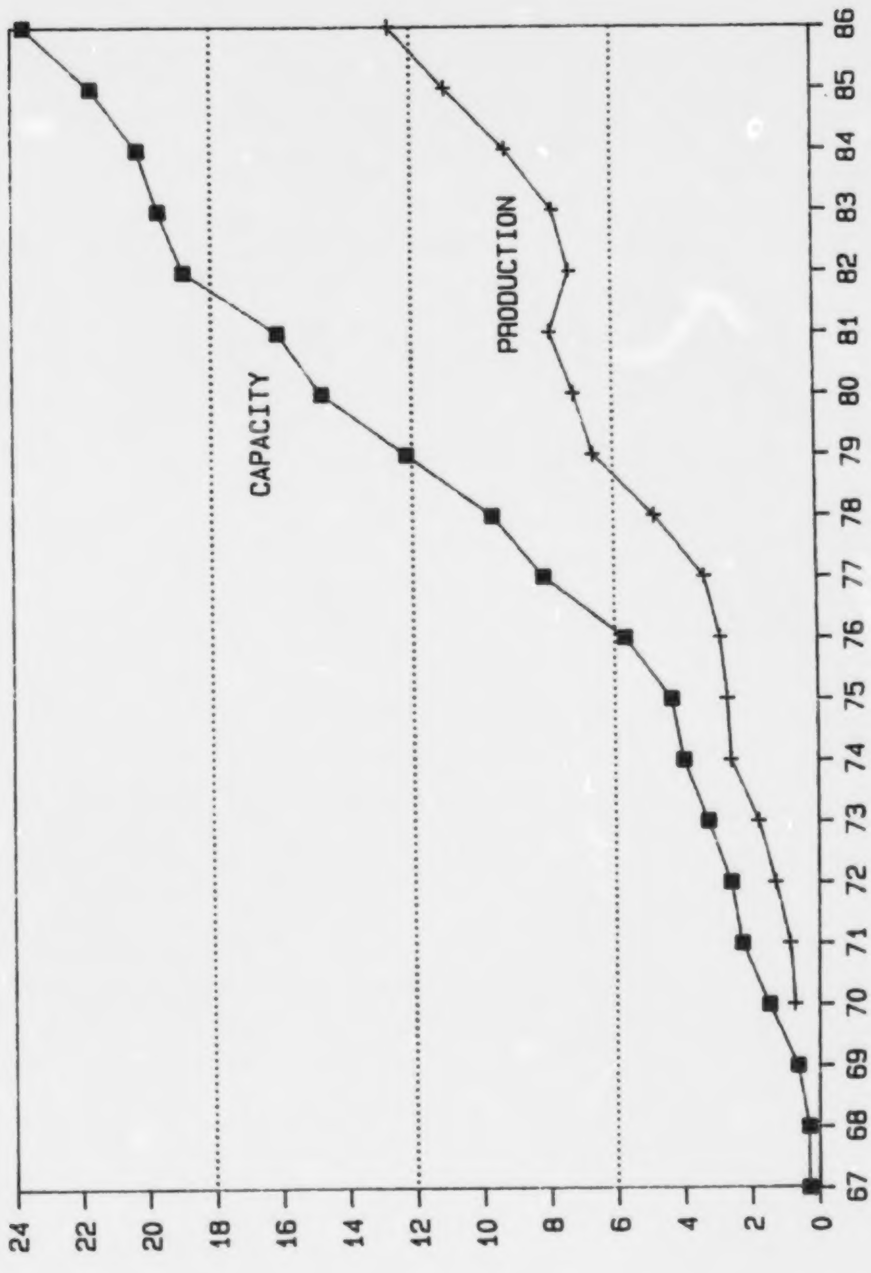

FIGURE 1

DRI PRODUCTION BY PROCESS

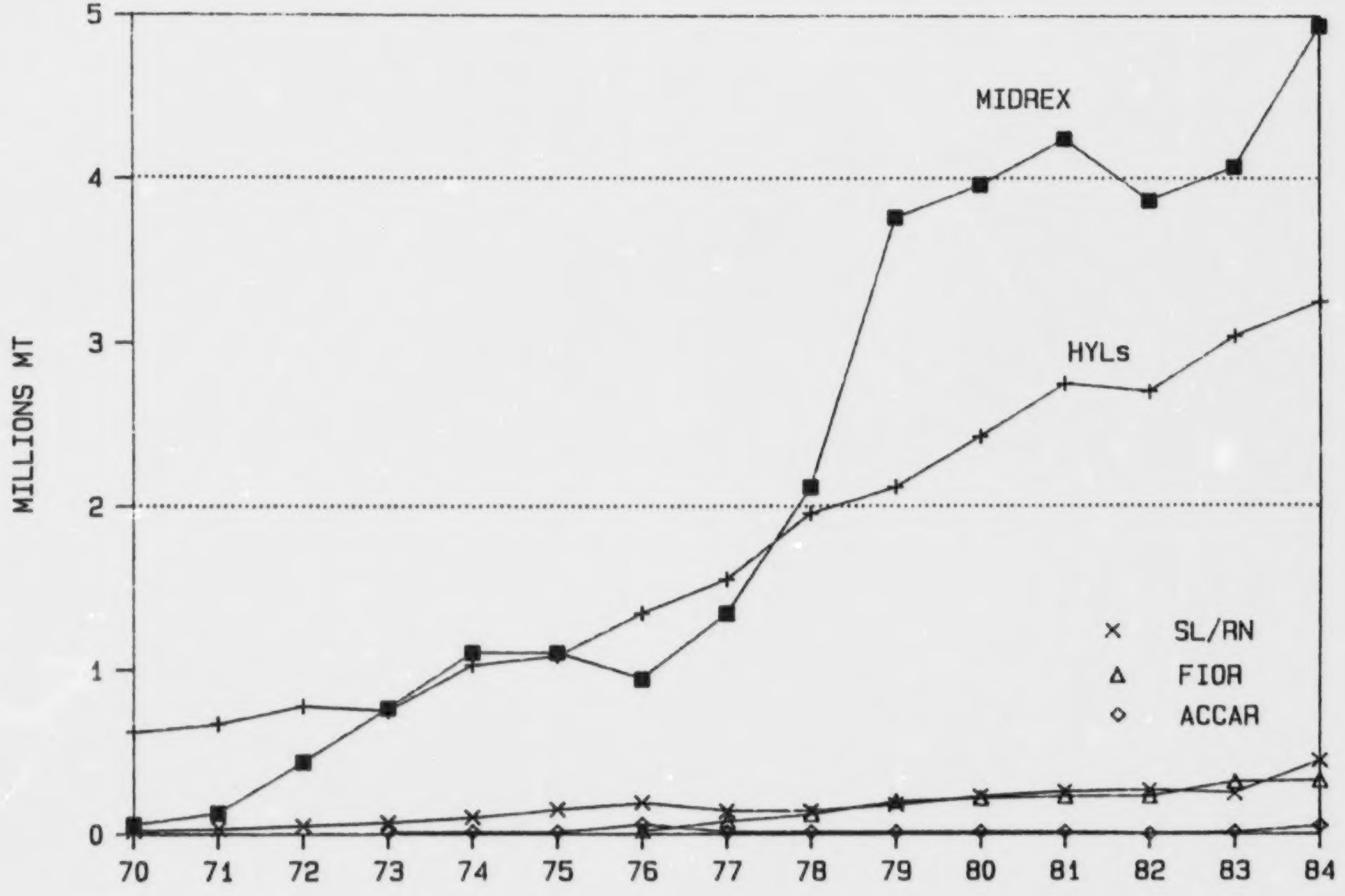

FIGURE 2

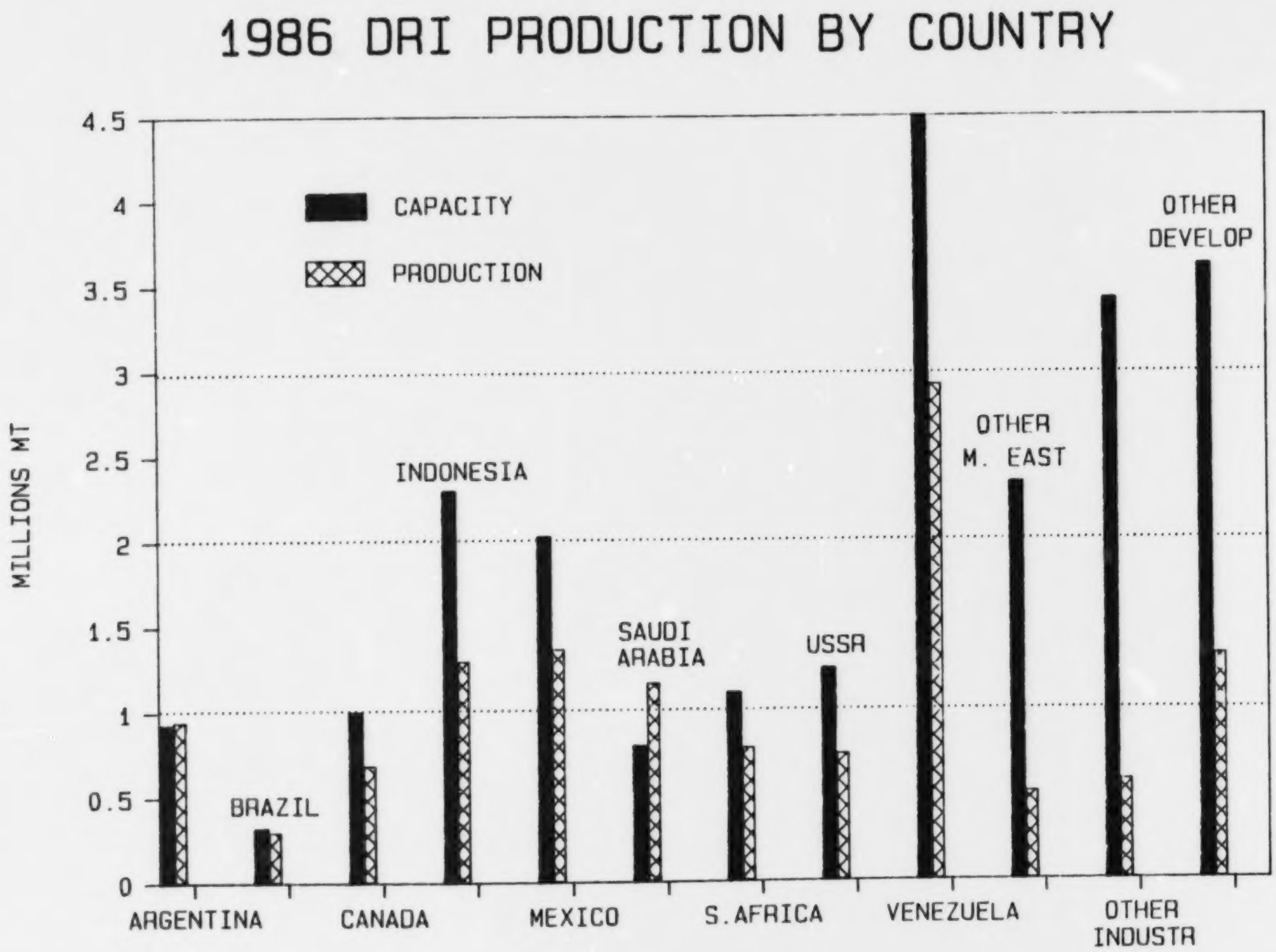
1986 DRI PRODUCTION BY COUNTRY
CAPACITY
PRODUCTION
MILLIONS MT
4.5
4
3.5
3
2.5
2
1.5
1
0.5
0
ARGENTINA
BRAZIL
CANADA
INDONESIA
MEXICO
SAUDI ARABIA
S.AFRICA
USSR
VENEZUELA
OTHER M. EAST
OTHER INDUSTR
OTHER DEVELOP

FIGURE 3

The forecasts for the world production of DRI for 1985, which had been made in the mid-to-late 1970's, were for 50 million tpy. The actual production in 1985 was only ¼ of these earlier forecast production rates.

Possible reasons for this lower total DRI production are the following changes:

(a) Lower steel demand.
(b) Lower scrap steel prices.
(c) Higher natural gas prices.

The historical accuracy of forecasts on DRI production should be borne in mind in evaluating the potential long term future of the LB process.

The first direct reduction plant to be used exclusively to manufacture merchant DRI was a Midrex installation commissioned in 1981 at Emden, West Germany with a 880,000 tpy capacity. In anticipation of a future need for larger capacity space was set aside for additional modules. This plant is currently idle. The plant initially used Brazilian ore pellets.

Midrex Corporation estimated the reasons for 1986 unused capacity as:

Limitations due to Technical Problems	= 18.2% unused
Market Factors	= 15.8% unused
Unused Capacity started-up in 1986	= 7.1% unused
Idled due to War	= 5.2% unused
TOTAL UNUSED CAPACITY	= 46.3% unused

Major DRI facilities under construction for the near future include:

LOCATION	CAPACITY (tpy)	PROCESS	STATUS
Sicartsa, Mexico	2.0 million	HYLIII	Delayed
NISCO, Mobarakeh, Iran	3.2 million	Midrex	1988 Start-up
I & S complex Misurata, Libya	1.1 million	Midrex	Delayed
NISCO, Ahwaz, Iran	1.8 million	Midrex, HYL	Delayed
Soidac, Iraq	1.0 million	HYL	Delayed
OEMK II, USSR	3.2 million	Midrex	1995 ? (planned)

6.0 COMPARISON OF EXISTING & UPCOMING PROCESSES

6.1 Categorization by Feed Materials

An examination of the attributes of the existing and developmental processes has been conducted. Table 1 shows a breakdown of major processes by type of ore feed (fines vs. pellet/sinter), reductant energy form (coal vs. coal + coke vs. gas) and product (sponge iron vs. liquid iron). Most of these processes also require electricity for melting. A process which uses coal for melting offers advantages in developing countries with coal resources.

Technologies using pellets/sinter are the most common, especially when using gas for production of DRI. Only the Elred, Plasmasmelt, FW-Tetronics plasma and Inred processes use ore fines and coal reductant; they would therefore be direct competitors to the LB. In terms of reductant energy form and ore feed, the LB process fills a need.

There are three other processes, which have similarities to the LB with respect to heating. However, neither is significant today. The Echeverria process used indirect heating based on pellet/sinter feed and 100% coal as a reductant. The only plant, with a capacity of 20,000 tpy, was in Spain. Plans for plants with capacities up to 250,000 tpy never materialized. The Kinglor-Metor process utilizing pellets/ore and coal have externally heated silicon carbide reactor shafts. The original plants in Italy are closed. Two 20,000 tpy plants in Burma are said to be operating. The Hoganas process, also used indirect heating along with fine ore feed and 100% coal reductant. These processes have never been used for production of steelmaking grade DRI in a major facility.

Conceptually, it is feasible to produce DRI using gasified coal as the reductant. The Westinghouse Integrated Steelmaking process is based on this. Other established natural gas based DRI processes such as Midrex could be used in conjunction with a coal gasification plant. The coal has a significantly lower heating value than natural gas, requires larger equipment and the cost of the coal gasifier must be considered. Due to its relatively higher capital and operating costs there are no plants based on this technology and they are unlikely in the near future.

Kawasaki is also developing a liquid ironmaking process - the XR process - based on ore fines and low-grade coke (Reference 11). The process consists of two fluid-bed chambers: a main smelting-reduction furnace and an auxiliary ore-reduction furnace. Kawasaki claims that the conversion of an existing blast furnace to the XR process is relatively simple, and is considering the conversion of one of its older blast furnaces. They are planning the construction of a 100 tpd demonstration plant within 5 - 6 years.

An enhanced form of DRI is merchant hot briquetted iron (the Midrex product is known as HBI). It is dense (almost twice as dense as lump DRI), uniformly sized, highly reistant to oxidation and easier to ship, handle and store. The International Maritime Organization has granted preferential guidelines for shipping of HBI. The world's first Midrex plant incorporating a hot briquetting system was started up in 1984 at Sabah Gas Industries, Malaysia. Midrex HBI has been used at steelplants in several countries including Japan, Spain, Turkey and India.

TABLE 1

CLASSIFICATION OF MAJOR DRI/LIQUID IRON PROCESSES

		REDUCTANT ENERGY FORM		
		COAL ONLY	COAL / COKE	GAS
ORE FEED	PELLETS/SINTER	. SL/RN (DRI) . ACCAR (DRI) . MIDREX EDR (DRI) (needs elec.) . KR (LI) . COIN (LI) . KRUPP CODIR (DRI) . COMBISMELT (LI)	. COKE OVENS/ BLAST FCES (LI) . SUMITOMO SC (LI) . ELECTRIC FURNACE SMELTING	. MIDREX (DRI) . HYL III (DRI) . ACCAR (DRI) . ARMCO (DRI) . PUROFER (DRI)
	FINES	. ELRED (LI) . LB (DRI/LI) . FW - TETRONICS PLASMA (STEEL)	. PLASMASMELT (LI) . INRED (LI)	. FIOR (DRI)

Notes: (1) The product from each of the processes is given in brackets.
LI refers to Liquid Iron

6.2 Categorization by Reactor Type

The three most common reactors for direct reduction are:

(1) Vertical Shaft The ore continuously moves down the shaft by gravity, and the product exits at the bottom. The flow between the ore and gases is generally countercurrent.

(2) Rotary Kiln The ore and solid reductant (if any) is continuously charged to the upper end of the inclined kiln. The flow between the ore and gases is generally countercurrent.

(3) Fluidized Bed The ore is reduced in a fluidized bed. The size of the ore is finer than that required for the other processes.

Table 2 is a breakdown of major DRI processes by type of reductant and reactor type (shaft vs. kiln vs. fluidized bed vs. indirect heating). The LB process has been classified as "indirect heating" because of the use of a heating tube although there is direct radiation heating at the product exit. There are no processes, gas or coal based, which use a screw-conveyor in the reactor vessel as proposed for the LB.

Table 2 also gives a breakdown of liquid ironmaking processes. Again, there are no processes which use indirect heating or a screw conveyor.

6.3 Product Characteristics

Typical characteristics of the LB and other processes at the sponge iron stage are:

	LB	TYPICAL FOR OTHER PROCESSES
Carbon Content:	1%	1 - 2%
Metallization:	80 - 90%	85 - 95%
Iron Yield:	98 - 99%	98 - 99%

There are no significant differences. However, a 80% metallized sponge, (a possibility suggested by the inventors), would not be economical as a feed for electric arc furnaces.

TABLE 2

CLASSIFICATION OF MAJOR DRI/LI PROCESSES BY REACTOR TYPE

100% COAL REDUCTANT	. MIDREX EDR	. SL/RN . KRUPP CODIR . ACCAR		. LB
GAS REDUCTANT	. MIDREX . HYL III . ARMCO . PLASMARED . PUROFER	. ACCAR	. FIOR	
	SHAFT	KILN	FBR	INDIRECT HEATING
	REACTOR TYPE			

CLASSIFICATION OF MAJOR LI PROCESSES BY REACTOR TYPE

COAL/COKE REDUCTANT	. BLAST FCE . COIN . KR	.COMBISMELT (Steelmaking)	. INRED . ELECTRIC SMELTING FCE.	. ELRED . PLASMASMELT	. LB
GAS REDUCTANT					
	SHAFT	KILN	FLASH/SMELTING FCE	FBR	INDIRECT HEATING
	REACTOR TYPE				

6.4 Spreadsheet Comparison

Table 3 is a comparison of key technical parameters and overall economics of various DRI and liquid ironmaking processes - both developed and developing.

The installed capacities and state of development of each process is listed to show overall status. As discussed earlier, there are numerous processes but Midrex and HYL have 90% of the market. The ore feed size and reductant type requirements are given. For the liquid ironmaking processes, the energy form (coal, electricity, etc.) required for the melting operation and vessel type is listed; for the sponge iron processes, it is assumed that melting is done in an EAF for production of steel.

The energy exports are the by-product energy forms (fuel, steam, electricity) which are produced and can possibly be used elsewhere. The specific energy consumptions to the liquid steel stage are listed. These figures are a guide only, as the basis for the different processes is not always fully consistent. For the DRI, but not the LI processes, the breakdown of the energy consumption between the sponge iron production and steelmaking stages is listed. The conversion of electrical energy units from kwh to Gcal has been on a factor of 859 kcal/kwh (equivalent to 3412 BTU/kwh). An alternative approach, if the primary energy requirement for production of electricity is to be considered, would be to allow for a 35 - 40% conversion efficiency (based on a steam power plant) at the electric power plant. This approach would significantly increase the total specific energy consumption for those processes which are electricity intensive. The net total specific energy consumption figures include a credit for by-product energy. The actual net total consumptions are lowest for the Midrex, HYLIII, Coke Oven/Blast Furnace/BOP processes. The given number for Plasmared and Midrex EDR are lower, but they are based on calculations or pilot plant facilities and not commercial processes. The Elred and Combismelt process have a net requirement in the range of 4.5 - 5.3 Gcal/t. The Combismelt process requires no external electricity as all electricity is internally generated within the plant.

Operating costs (excluding taxes, financing) and capital costs per tonne liquid steel are given for a U.S. site. To be competitive, the costs for the LB process would have to be no greater than U$ 220-410/Atls Capital and U$ 140-220/tls operating.

TABLE 3 - SPREADSHEET OF MAJOR PROCESSES

PROCESS	PRODUCT	LICENSOR	STATE OF DEVELOPMENT	INSTALLED CAPACITY 1986 TOTAL (million tpy)	INSTALLED CAPACITY RATED LARGEST SINGLE UNIT (thousand tpy)	ORE FEED SIZE	REDUCTANT	REDUCING REACTOR	MELTING VESSEL	ENERGY FOR MELTING	ENERGY EXPORTS	SPECIFIC ENERGY CONSUMPTION (1) DRI STAGE (Fuel-Gcal/t) (Elec-kWh/t)	SPECIFIC ENERGY CONSUMPTION (1) FOR STEELMAKING (kWh/t)	SPECIFIC ENERGY CONSUMPTION (1) NET TOTAL (Gcal/tls)	OPERATING COST (US$/tls) (2)	CAPITAL COST (US$/Atls) (2)
COKE OVEN & BLAST FCE	LIQUID IRON		Fully Commercialized	900	3,500+	PELLETS	COKE	SHAFT	SHAFT	COKE (possibly NG/OIL/COAL)	FUEL	-	(BOP)	3.3	160	315
MIDREX	DRI	KOBE STEEL	Fully Commercialized	13.3	600	PELLETS, LUMPS	NG/OIL	SHAFT	-	-	NO	2.8 110	715 (EAF)	3.7	220	225
HYLIII	DRI	HYLSA	Fully Commercialized	2.8 7.8(HYL I)	500	PELLETS, LUMPS	NG/OIL	SHAFT	-	-	NO	2.8 110	715 (EAF)	3.7	220	225
ARMCO	DRI	ARMCO STEEL	Abandoned	0	-	PELLETS, LUMPS	NG	SHAFT	-	-	NO	3.1 40	715 (EAF)	4.0	220	220
PLASMARED	DRI	SKF	Pilot Plant	0	-	PELLETS	NG/OIL	SHAFT	-	-	NO	1.4 860	715 (EAF)	3.0	?	?
MIDREX EDR	DRI	KOBE STEEL	Pilot Plant	0	-	PELLETS, LUMP	COAL	SHAFT	-	-	FUEL	2.5(0.8) 860	715 (EAF)	3.3	225	310
SL/RN	DRI	LURGI	Commercialized	0.99	165	PELLETS, LUMPS	COAL	KILN	-	-	STEAM	5.6(1.8) 85	715 (EAF)	4.9	190	275
ACCAR	DRI	ALLIS CHALMERS	Commercialized	0.15	150	PELLETS, LUMPS	OIL/GAS/ COAL	KILN	-	-	STEAM	4.4(1.0) 85 (80% coal)	715 (EAF)	4.4	195	245
FIOR	DRI	DAVY McKEE	Commercialized	0.4	400	FINES, PELLETS, LUMPS	NG/OIL	FLUID BED		-	NO	4.2 65	715 (EAF)	5.2	205	275
COREX	LIQUID IRON	KORF/VOEST-ALPINE	First commercial plant under construc.	0	-	PELLETS	COAL	SHAFT	FB	COAL	NO	-	(BOP)	4.1	?	?
INRED	LIQUID IRON	BOLIDEN	Abandoned	0	-	FINES	COAL	FLASH	EF	ELEC.	FUEL	-	(BOP)	4.4	145	360
ELRED	LIQUID IRON	ASEA	Pilot Plant	0	-	FINES	COAL	FB	EF	ELEC.	ELEC.	-	(BOP)	4.5	140	410
COMBISMELT	LIQUID IRON	LURGI/DEMAG	Commercialized (NZS)	0	-	PELLETS, LUMP FINES	COAL	KILN (SL/RN)	SEF	ELEC. (INTERNAL)	ELEC.& STEAM	-	(BOP)	5.3	?	?
PLASMA-SMELT	LIQUID IRON	SKF	Pilot Plant	0	-	FINES	COAL	FB	SHAFT	ELEC.	NO	-	(BOP)	4.7	160	270
LB	LIQUID IRON		Laboratory scale testwork	0	-	FINES	COAL	INDIRECT HEATING	EF	ELEC.	FUEL	-	(BOP)	?	?	?

NOTES: 1) Net total specific energy consumption is per tonne liquid steel (tls).
Figures generally taken from Reference 3. The figures for DRI stage are gross requirements per tonne DRI, the values in brackets are the energy credits if any. Net total specific energy consumption for DRI processes = Net energy DRI stage x 1.087 metallization factor + steelmaking energy. Electricity units conversion factor has been taken as 869 kcal/kWh (3412 BTU/kWh). Oxygen is considered to be a purchased material and its energy requirements are excluded. Scrap charge to EAF is approximately 25% of total charge.

2) Costs are based mostly from Reference 3 which gives 1981 costs for a U.S. site. The costs listed are 1987 dollars and include a 10% escalation for inflation. Atls is annual tonne liquid steel. Operating cost includes a credit for by-product energy production.

6.5 Comparison with Significant Competing Processes

Combismelt Process

This process is a combination of the SL/RN kiln and submerged electric arc furnace for production of iron. The feed materials are pellet/sinter ore and coal. It includes an energy recovery plant for the waste gases to produce electricity for use in the electric arc furnace. The product can be adjusted to give low or high carbon content. It is claimed that the low carbon metal can be finished in a ladle to obtain any steel grade.

There are currently no facilities using the COMBISMELT process. However, the 750,000 tpy steel plant at New Zealand Steel (Glenbrook), which is the predecessor of the COMBISMELT, is a proven demonstration of the components of this technology (Reference 8). The New Zealand steel plant has recently been expanded to a capacity of 750,000 tpy steel including 4 kiln units for direct reduction.

The iron ore used at this plant is an ironsand mineral. The ironsand and coal is charged to a multi-hearth furnace which drives off most of the volatiles. There were major gas handling constraints with the initial configuration which excluded a multi-hearth furnace. The hot ironsand/char mixture is directly fed to the rotary kiln by a pan conveyor. The kiln product, at about 1000°C , 5%C and over 85% metallization is captured continuously in insulated containers for transfer to the electric melting furnaces. These furnaces are rectangular with six electrodes each. The target electricity consumption is 850 kwh/t, but the initial results report 1100 kwh/t. The hot metal is finished to steel in a K-OBM furnace. The installation of a waste heat recovery system to produce electricity from waste off-gas is planned.

The chemical analysis and sizing of the ironsand (Reference 6) is:

COMPOSITION

Fe_t = 58.0%

Moisture = 5.0%
CaO = 1.0%
SiO_2 = 3.7%
TiO_2 = 8.0%
AL_2O_3 = 4.2%
MgO = 3.0%

SIZE

+ 300 microns = 1%
212 = 2%
150 = 20%
106 = 49%
75 = 25%
53 = 2%
-53 = 1%

The coal chemical and size analysis (Reference 6) is:

COMPOSITION

Moisture = 20.0%

Volatiles = 42.5% (dry basis)
Ash = 6.0% (dry basis)
Fixed Carbon = 51.5% (dry basis)

SIZE

+ 13.2 mm = 30%
9.5 = 16%
6.7 = 15%
3.5 = 14%
1.0 = 17%
- 1.0 = 8%

The coal char leaving the multiple-hearth furnace contains 10% volatiles.

Corex Process

This process is a two stage, coal based liquid ironmaking process in a relatively advanced state of development. The developers (Korf-Voest-Alpine) have a 60,000 tpy demonstration facility located in Kehl, West Germany. Ore is reduced in a shaft furnace and then fed directly to a fluidized bed melter-gasifier. The energy for melting is provided by combustion of coal with oxygen. The coal can be any type, with size up to 1½ inch, compared with the fines used for LB. The design operating pressure of the melter-gasifier unit is 5 bars; this high pressure may possibly lead to operating problems for a large production unit.

A major commercial Corex plant, at 300,000 tpy capacity, is to begin operation at ISCOR, South Africa in 1988. Recently, Weirton Steel cancelled plans to construct a 300,000 tpy Corex plant after negotiations with Korf Engineering, lasting over a year, had failed.

7.0 OPERATING EXPERIENCE OF EXISTING PROCESSES

In assessing the potential of the LB process, it is important to review the operating experience of some of the DRI/liquid ironmaking processes. An understanding of any limiting technical factors which have restricted the applications of these processes could give an indication of the potential relative success of the LB process.

The gas-based DRI processes which use a shaft reactor (Midrex, HYL) are well developed and well proven in operation in commercial plants., Single Midrex units have achieved production in excess of 600,000 tpy. The Armco & Purofer processes (reducing shaft and gas-based), are not in active production and are no longer being marketed.

The FIOR process, (using four fluid bed reactors in series), has had some technical difficulties. The bed can collapse if the particle size distribution is not uniform or because of sintering of the fine ore. The formation of liquid material closes off the transfer ducts between the reactors. The technology is also not being actively marketed.

The SL/RN and ACCAR kiln processes have problems of "ringing" of the kiln shell with unit sizes above about 250,000 tpy. The softening of the gangue constituents and partially reduced ore occurs due to the high temperatures required for heat transfer.

The Inred process had numerous problems at the pilot plant stage and has been abandoned. The problems were largely due to the attempt to do both smelting and melting in the same vessel. Control of the heat distribution was difficult, and large quantities of waste heat steam had to be handled.

The Elred, Corex Plasmared and Plasmasmelt processes have not reported major operating difficulties to date. However, experience with these processes has been at a developmental level, and not with a significant commercial facility.

8.0 TECHNOLOGICAL TRENDS

There are numerous trends in the iron and steel industry which could impact the success of the LB process. Some of these are described below.

For high tonnage iron production, there is no major existing alternative to the blast furnace and there are no upcoming technologies in the near future which would challenge it.

The injection of pulverized coal (or other fuels) directly into the blast furnace is an effective means of reducing coke rate and fuel costs. Typical replacement ratios (kg coal/kg coke) are reportedly in the range of 0.9 - 1.1. Coal injection is now widely practiced with injection rates of up to 100 - 150 kg/thm. Typical coke rates in blast furnaces without coal injection are 400 - 500 kg/thm. Coal injection can increase the cost competitiveness of blast furnaces.

For lower tonnage facilities, there is much development in the area of coal based ironmaking. A Corex process commercial plant is currently being built at ISCOR, South Africa. The plant capacity will be 300,000 tpy. Assuming that the experience with this plant indicates that higher capacity plants are technically feasible, it will likely be another decade before this process can be scaled up to a 1 million tpy level.

Other technologies (Elred, Combismelt, Plasmasmelt, etc.) are being actively marketed by their proponents. Plasma technologies have not penetrated the iron and steel industry to the extent earlier forecast. The reasons may be related more to the economics, particularly power costs, rather than the technical feasibility.

The development of thin strip casting at a reasonable capacity level would be a major revolution in the steel industry. It could make the production of flat products economical at a possible minimum level of 250-300,000 tpy. The production of flat steel which is to undergo deep drawing requires low levels of tramp elements, often requiring virgin iron rather than scrap steel. Therefore, there could be an increased demand for virgin iron units at lower tonnage levels than that provided by blast furnaces.

9.0 POTENTIAL COMPETITIVENESS OF THE LB PROCESS

The purpose of this review is to ascertain the potential role the LB could attain for the production of virgin iron. As there are no existing fully commercialized ironmaking processes based on coal and ore fines, a successful process with good raw material and energy economics would fill a desirable technological niche. The benefits anticipated with the use of lower cost coal (where available) without the necessity for coking and ore fines without sintering have to be balanced against the overall efficiency of the reductant energy use and the cost of electrical power for the melting of the LB produced sponge iron. Also, the costs associated with the construction, operation and maintenance of the process would have to compare favourably with those similar costs for competitive processes.

The technical review of the LB process idea is given in two other reports within this current study. Without repeating these reports, it is sufficient to say that several major considerations appear to remain and to require further fundamental investigations, i.e.:

- the optimization of the use of the reductant energy with efforts made to identify and overcome heat transfer limitations.

- once a firm design approach has been selected then it will be important to solve all structure requirements for the complexity and temperature design features which are necessary for the operation of the process reactors.

- the operation in a commercial plant will require attention given to the critical need for an effective solids transport system.

Even without this critical information, the current evaluation suggests the LB process would have a minimum unit throughput limit which would not allow it to compete with the blast furnace. The most likely area in which it can compete for ironmaking is against DRI and related liquid iron processes up to an upper size limit of between 200,000 and 400,000 tpy per unit. Even this production capacity appears difficult to achieve with the indirect heating approach used in the LB.

From the standpoint of currently available technology, it must be established if the major considerations given above (heat transfer, structural, and solids handling) can be mastered economically. The process operability of the LB process, as currently envisaged, is unknown. Steel plants require systems which can withstand the severe environment, operate continuously without significant downtimes, and can be well controlled. Significant experimental and developmental work will be required to develop, if at all feasible, an LB process plant which can operate with a high availability.

Table 4 is an analysis of the relative potential competitiveness of the LB process. Qualitative comparisons are made with the Midrex, SL/RN and FIOR processes for sponge iron production and Blast Furnace, Elred, Corex and Combismelt for liquid iron production. These other processes represent established technologies or major developing technologies.

For areas which are abundant in coal, LB is equal to or better than the other processes from an energy standpoint (as it uses coal rather than natural gas or coke). The Combismelt process has the advantage that the electricity required for melting is generated as a by-product.

The LB process has an advantage in ore feed preparation requirements over processes which need pellets or sinter. However,the handling of fine ores may present difficulties for some plants.

There are major uncertainties at this time regarding the capital, operating costs and process operability of the LB process. It is not clear if the LB can be competitive on these factors since the process is not sufficiently defined. Compactness and simplicity of equipment, ease of operation are desirable features for any new process.

The LB process will likely be superior in environmental factors compared to most other processes as the heat is provided indirectly which should result in low carry-over of the feed materials to the off-gas. In terms of maximum unit throughput, there appear to be significant limitations on the LB process.

The closest existing process to the LB in terms of ore size and reductant is the New Zealand Steel operation based on the SL/RN process. This process is at least 10 - 15 years ahead of the LB process in the state of development.

For the range of throughput capacities envisaged for the LB process, this process may be most suitable for the developing countries which require a small scale steelmaking facility. However, it is important to reflect on the fact that approximately 18% of the DRI capacity in 1986 was idled due to technical problems. Many of these problems occur in the developing countries, emphasizing their need for technologies which can be operated using the local technical infrastructure. Therefore, for the LB process to be successful in developing countries, the plant should not be overly complex.

TABLE 4

RELATIVE POTENTIAL COMPETITIVENESS OF LB PROCESS

	FOR SPONGE IRON PRODUCTION			FOR LIQUID IRON PRODUCTION			
				HIGH TONNAGE	LOW TONNAGE		
	MIDREX	SL/RN	FIOR	Coke Oven/ Blast Furnace	Elred	KR	COMBISMELT
1) Use of coal for Energy	+	=	+	+	=	=	-
2) Specific Energy Consumption	?	?	?	?	?	?	?
3) Ore Feed Preparation	+	+	=	+	=	+	?
4) Ore Feed Handling	?	?	=	?	?	?	?
5) Environmental Considerations	=	+	+	+	+	+	+
6) Non-Energy Operating Costs	?	?	?	?	?	?	?
7) Capital Costs (Incl. Raw Material Prepn.)	?	?	-	?	-	?	?
8) Maximum Unit Throughput	-	=	-	Strongly -	-	-	=

NOTES: 1. A Positive (+) means LB process is superior.

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NOTE: The above is a selected list of literature used in the preparation of this report. Numerous others were also reviewed but are not listed.

APPENDIX - LIST OF DRI PLANTS & PRODUCTION

Direct Reduction Plants Producing Steelmaking-Grade DRI

(Installed or Under Construction as of June 1, 1987)

Start Up Year	Plant	City	Country	Process	No. Of Units	Type Of Fuel	Capacity (MTPY)	Status
1957	Hylsa 1M	Monterrey	Mexico	HyL I	1	Gas	0.10	Idle
1960	Hylsa 2M	Monterrey	Mexico	HyL I	1	Gas	0.27	Converted
1967	Tamsa	Veracruz	Mexico	HyL I	1	Gas	0.24	Operating
1969	OSM	Portland	USA	Midrex	2	Gas	0.30	Dismantled
1969	Hylsa 1P	Puebla	Mexico	HyL I	1	Gas	0.31	Closed
1970	Thyssen	Oberhausen	W. Germany	Purofer	1	Gas	0.15	Dismantled
1970	NZS 1	Glenbrook	New Zealand	SL/RN	1	Coal	0.17	Operating
1971	NHSW	Hamburg	W. Germany	Midrex	1	Gas	0.40	Operating
1971	GSC	Georgetown	USA	Midrex	1	Gas	0.40	Operating
1972	Armco	Houston	USA	Armco	1	Gas	0.33	Closed
1973	NML	Niagara Falls	Canada	Accar	1	Gas/Coal	0.04	Closed
1973	K-M	Buttrio	Italy	K-M	1	Coal	0.01	Closed
1973	Sidbec 1	Contrecoeur	Canada	Midrex	1	Gas	0.40	On Standby
1973	Piratini	Charquedas	Brazil	SL/RN	1	Coal	0.07	Operating
1973	Dunswart	Benoni	S. Africa	Codir	1	Coal	0.15	Operating
1974	Hylsa 3M	Monterrey	Mexico	HyL I	1	Gas	0.48	Converted
1974	Usiba	Bahia	Brazil	HyL I	1	Gas	0.25	Operating
1975	Stelco	Red Lake	Canada	SL/RN	1	Coal	0.35	Closed
1976	Arved	Cremona	Italy	K-M	1	Coal	0.04	Dismantled
1976	SMC	Falconbridge	Canada	Accar	1	Gas/Oil	0.24	Dismantled
1976	SIDERCA	Campana	Argentina	Midrex	1	Gas	0.33	Operating
1976	SIDOR I	Matanzas	Venezuela	HyL I	1	Gas	0.36	Operating
1976	Fior	Matanzas	Venezuela	Fior	1	Gas	0.40	Operating
1977	Cosigua	Santa Cruz	Brazil	Purofer	1	Oil	0.35	Dismantled
1977	NSC	Hirohata	Japan	NSC	1	Kerosene	0.15	Closed
1977	SIDOR	Matanzas	Venezuela	Midrex	1	Gas	0.35	Operating
1977	Sidbec 2	Contrecoeur	Canada	Midrex	1	Gas	0.60	Operating
1977	Hylsa 2P	Puebla	Mexico	HyL I	1	Gas	0.63	Operating
1977	NISCO	Ahwaz	Iran	Purofer	1	Gas	0.33	Idle
1978	DRC	Rockwood	USA	DRC	1	Coal	0.06	Closed
1978	ACINDAR	Villa Const	Argentina	Midrex	1	Gas	0.60	Operating
1978	QASCO	Umm Said	Qatar	Midrex	1	Gas	0.40	Operating
1978	Krakatau	Kota Baja	Indonesia	HyL I	1	Gas	0.58	Operating
1979	SIDOR	Matanzas	Venezuela	Midrex	3	Gas	1.28	Operating
1979	BSC	Hunterston	UK	Midrex	2	Gas	0.80	Idle
1979	Soidac	Khor Al-Zubair	Iraq	HyL I	2	Gas	0.49	Idle
1979	Hylsa 2M5	Monterrey	Mexico	HyL III	1	Gas	0.25	Operating
1980	ISCOTT	Point Lisas	Trinidad	Midrex	1	Gas	0.42	Operating
1980	Krakatau	Kota Baja	Indonesia	HyL I	1	Gas	0.57	Operating
1980	SIDOR II	Matanzas	Venezuela	HyL I	2	Gas	1.41	Operating
1980	Siderperu	Chimbote	Peru	SL/RN	3	Coal	0.10	Operating
1980	SIIL	Paloncha	India	SL/RN	1	Coal	0.03	Operating
1981	SKF	Hofors	Sweden	Plasmared	1	Gas	0.07	Closed
1981	Nordferro	Emden	W. Germany	Midrex	2	Gas	0.88	Idle
1981	SIDOR II	Matanzas	Venezuela	HyL I	1	Gas	0.70	Operating
1981	Mining Corp	Maymo	Burma	K-M	1	Coal	0.02	Operating
1982	Hadeed	Al-Jubail	Saudi Arabia	Midrex	1	Gas	0.40	Operating
1982	ISCOTT	Point Lisas	Trinidad	Midrex	1	Gas	0.42	Operating
1982	DSC	Warri	Nigeria	Midrex	2	Gas	1.02	Operating
1982	Krakatau	Kota Baja	Indonesia	HyL I	2	Gas	1.15	Operating
1983	Hadeed	Al-Jubail	Saudi Arabia	Midrex	1	Gas	0.40	Operating
1983	OEMK	Kursk	USSR	Midrex	1	Gas	0.42	Operating
1983	HYLSA 3M5	Monterrey	Mexico	HyL III	1	Gas	0.50	Operating
1983	OSIL	Keonjhar	India	Accar	1	Coal	0.15	Operating
1983	Scaw Metals	Germiston	S. Africa	DRC	1	Coal	0.08	Operating
1984	SGI	Labuan	Malaysia	Midrex	1	Gas	0.72	Operating
1984	ISCOR	Vanderbijlpark	S. Africa	SL/RN	4	Coal	0.60	Operating
1984	SIIL	Paloncha	India	SL/RN	1	Coal	0.03	Operating
1984	Mining Corp	Maymo	Burma	K-M	1	Coal	0.02	Operating
1985	OEMK	Kursk	USSR	Midrex	1	Gas	0.41	Operating
1985	NISCO	Ahwaz	Iran	Midrex	1	Gas	0.40	Idle
1985	Perwaja	Chuka	Malaysia	NSC	1	Gas	0.60	Closed
1985	Davsteel	Vanderbijlpark	S. Africa	DAV	1	Coal	0.03	Operating
1985	USCO	Vereenging	S. Africa	USCO	1	Gas	0.25	Closed
1986	OEMK	Kursk	USSR	Midrex	1	Gas	0.42	Operating
1986	ANSDK	El Dikheila	Egypt	Midrex	1	Gas	0.72	Operating
1986	NZS 2	Glenbrook	New Zealand	SL/RN	4	Coal	0.70	Operating
1986	Ipitata	Joda	India	Tisco	1	Coal	0.09	Operating
1987	OEMK	Kursk	USSR	Midrex	1	Gas	0.41	Start 3QTR
1987	BSIL	Chandil	India	SL/RN	1	Coal	0.15	Under Cons
1988	Sunflag	Bhandara	India	Codir	1	Coal	0.15	Under Cons
1988	Sicartsa	Las Truchas	Mexico	HyL III	4	Gas	2.00	Delayed
1988	NISCO	Mobarakeh	Iran	Midrex	5	Gas	3.20	Under Cons
1988	I&S Complex	Misurata	Libya	Midrex	2	Gas	1.10	Delayed
1989	NISCO	Ahwaz	Iran	Midrex	2	Gas	0.80	Delayed
1989	NISCO	Ahwaz	Iran	HyL I	3	Gas	1.00	Delayed
1989	Soidac	Khor Al-Zubair	Iraq	HyL I	2	Gas	1.00	Delayed

"DIRECT FROM MIDREX" V12, n.4

DRI Market Developments

WORLDWIDE DRI PRODUCTION UP 15%

MIDREX Plants Produce 54% Of Total

Production of direct reduced iron rose to 12.68 million metric tons (tonnes) in 1986, an increase of 15 percent over 1985 worldwide DRI output.

The substantial increase in worldwide DRI production is attributed to 1) improved plant production by coal-based facilities in South Africa and gas-based modules in Venezuela, Indonesia, Saudi Arabia, Trinidad and the Soviet Union, and 2) start up of additional capacity in India and New Zealand (coal-based plants), Egypt and the Soviet Union (gas-based plants).

The installed rated capacity of all direct reduction processes increased 9 percent during 1986 to 23.62 million tonnes. (Table I)

Coal-based DR plants produced 1.18 million tonnes in 1986, up 46 percent from 0.81 million tonnes in 1985. Gas-based DR remained the dominant DR route producing 91 percent or 11.50 million tonnes, up 13 percent from 10.21 million tonnes in 1985.

An analysis of capacity utilization for 1986 is shown in Table II.

MIDREX® Direct Reduction Plants produced 6.79 million tonnes of the 1986 worldwide total, which represents a 13 percent increase over 1985.

MIDREX Plants produced 54 percent of the global supply of DRI in 1986. This was the eighth consecutive year in which MIDREX Plants have accounted for over 50 percent of the worldwide output of DRI.

Production exceeded rated capacity at the following MIDREX Plants in 1986: HADEED, Saudi Arabia (1.17 million tonnes); Sidbec-Dosco II, Canada (0.69 million tonnes); ACINDAR, Argentina (0.63 million tonnes); and QASCO, Qatar (0.49 million tonnes).

Annual production records were set by the following MIDREX Plants in 1986: SIDOR II, Venezuela; HADEED, Saudi Arabia; OEMK, Soviet Union; Sabah Gas Industries, Malaysia; and ISCOTT, Trinidad and Tobago.

1986 DRI CAPACITY AND PRODUCTION*

COUNTRY	CAPACITY (Mt/y)	PRODUCTION (Mt)
Argentina	0.93	0.95
Brazil	0.32	0.30
Burma	0.04	0.03 **
Canada	1.00	0.69
Egypt	0.72	0.03
India	0.30	0.17
Indonesia	2.30	1.30
Iran	0.73	0.00
Iraq	0.49	0.00
Malaysia	1.32	0.58**
Mexico	2.03	1.37
New Zealand	0.87	0.26
Nigeria	1.02	0.11
Peru	0.10	0.06
Qatar	0.40	0.49
Saudi Arabia	0.80	1.17
South Africa	1.11	0.79
Sweden	0.07	0.00
Trinidad	0.84	0.38
U.K.	0.80	0.00
U.S.A.	0.40	0.16
U.S.S.R.	1.25	0.75
Venezuela	4.50	2.92
W. Germany	1.28	0.17
TOTAL	23.62	12.68

*Steelmaking grade DRI ** Estimated

TABLE I

ANALYSIS OF 1986 DRI CAPACITY UTILIZATION

	MILLION TONNES	%
Actual Production	12.68	53.7
Capacity Idled Due to War	1.22	5.2
Capacity Idled Due To Market Factors	2.66	11.3
Unused Capacity Started Up in 1986	1.68	7.1
Operating Capacity Limited by Market Factors	1.07	4.5
Operating Capacity Limited by Technical Problems	4.31	18.2
TOTAL	23.62	100.0

TABLE II

"DIRECT FROM MIDREX", V12, n1

Worldwide DRI Production (1970–1984)

Country	1970	1971	1972	1973	1974	1975	1976	1977	1978	1979	1980	1981	1982	1983	1984
Argentina							0.03	0.29	0.40	0.85	0.78	0.83	0.89	0.95	0.91
Brazil				0.01	0.14	0.21	0.26	0.32	0.29	0.31	0.27	0.23	0.23	0.25	0.25
Burma												0.01	0.01	0.00	0.01
Canada				0.20	0.35	0.39	0.30	0.36	0.65	0.92	0.89	0.96	0.50	0.54	0.50
India											0.01	0.02	0.03	0.04	0.08
Indonesia									0.01	0.13	0.37	0.47	0.44	0.57	0.74
Iran									0.15						
Iraq											0.02				
Italy									0.01	0.01					
Japan								0.02	0.03						
Malaysia															0.04
Mexico	0.62	0.67	0.78	0.75	0.90	0.91	1.12	1.32	1.63	1.50	1.63	1.72	1.51	1.50	1.45
New Zealand	0.02	0.03	0.05	0.06	0.09	0.10	0.10	0.10	0.12	0.15	0.15	0.16	0.16	0.16	0.17
Nigeria													0.08	0.16	0.15
Peru											0.02	0.05	0.04	0.03	0.08
Qatar									0.08	0.32	0.42	0.46	0.45	0.39	0.50
S. Africa				0.03	0.06	0.07	0.09		0.08	0.11	0.12	0.12	0.10	0.08	0.27
Saudi Arabia													0.01	0.35	0.73
Sweden												0.01	0.03	0.02	0.02
Trinidad											0.02	0.19	0.24	0.28	0.24
USA	0.06	0.12	0.30	0.45	0.67	0.65	0.56	0.57	0.54	0.79	0.62	0.54	0.07		0.12
USSR														0.02	0.37
Venezuela							0.03	0.22	0.49	1.04	1.43	1.64	2.17	2.39	2.48
W. Germany	0.03	0.04	0.17	0.30	0.39	0.36	0.41	0.16	0.35	0.51	0.45	0.48	0.32	0.07	0.10
Total	**0.73**	**0.86**	**1.30**	**1.80**	**2.60**	**2.69**	**2.90**	**3.36**	**4.83**	**6.64**	**7.20**	**7.89**	**7.28**	**7.80**	**9.21**

Worldwide DRI Production by Country (million tpy)

Process	1970	1971	1972	1973	1974	1975	1976	1977	1978	1979	1980	1981	1982	1983	1984
Midrex	0.06	0.13	0.44	0.77	1.11	1.11	0.95	1.35	2.12	3.77	3.97	4.25	3.87	4.08	4.94
HyL I	0.62	0.67	0.78	0.75	1.03	1.09	1.35	1.56	1.96	2.12	2.33	2.48	2.52	2.71	2.86
HyL III											0.10	0.27	0.18	0.33	0.39
NSC								0.02	0.03	0.00	0.00	0.00	0.00	0.00	0.00
Purofer	0.03	0.03	0.03	0.04	0.06	0.06	0.07	0.10	0.23	0.02	0.00	0.00	0.00	0.00	0.00
Armco				0.13	0.23	0.20	0.17	0.10	0.12	0.21	0.21	0.24	0,07	0.00	0.00
Fior							0.02	0.08	0.12	0.20	0.22	0.23	0.23	0.32	0.33
Plasmared												0.01	0.03	0.02	0.02
KM									0.01	0.01	0.00	0.01	0.01	0.00	0.01
SL/RN	0.02	0.03	0.05	0.07	0.10	0.15	0.19	0.14	0.14	0.18	0.23	0.26	0.27	0.25	0.45
DRC									0.01	0.01	0.01	0.01	0.00	0.03	0.08
CODIR				0.03	0.06	0.07	0.09	0.00	0.08	0.11	0.12	0.12	0.10	0.05	0.08
ACCAR				0.01	0.01	0.01	0.06	0.01	0.01	0.01	0.01	0.01	0.00	0.01	0.05
Total	0.73	0.86	1.30	1.80	2.60	2.69	2.90	3.36	4.83	6.64	7.20	7.89	7.28	7.80	9.21
Cumulative	0.73	1.59	2.89	4.69	7.29	9.98	12.88	16.24	21.07	27.71	34.91	42.80	50.08	57.88	67.09
Gas Based	0.71	0.83	1.25	1.69	2.43	2.46	2.56	3.21	4.58	6.33	6.83	7.48	6.90	7.46	8.54
Coal Based	0.02	0.03	0.05	0.11	0.17	0.23	0.34	0.15	0.25	0.31	0.37	0.41	0.38	0.34	0.67

Worldwide DRI Production by Process (million tonnes)

"DIRECT FROM MIDREX", V10, N4

PROJECT REPORT

HATCH ASSOCIATES LTD. CONSULTING ENGINEERS
HATCH
ASSOCIATES

P.R. #5006.6
File #5006.2

STELCO - LB PROCESS EVALUATION

December 21, 1987

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EAF DUST TREATMENT

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If you disagree with any information contained herein, please advise immediately.

1.0 INTRODUCTION

This brief review completes the overall study to identify the potential for the LB process in the steel industry. The marketing reviews "World Steel Trends", "Review of DRI/Liquid Ironmaking Processes", and "Summary Review of World Energy Trends", plus this review on "EAF Dust Treatment" complement the technical reviews for a horizontal mode of the LB process and a vertical mode of the same basic process. The potential for the LB process for EAF dust treatment has been a secondary interest in the overall study. As no experimental data yet exist (a program has been planned at McMaster University by the inventors but, to-date, no experimental runs have been made to deal with the critical need of how to extract the zinc values) we have elected to only investigate the status of competing processes to determine the need for additional processes.

The treatment of industrial by-products is driven by two major incentives. The most socially acceptable is where solutions to by-product wastes result in economically viable secondary products which may be a raw material suitable for recyling or a by-product suitable for other uses. The other incentive is also driven by social needs, i.e. to remove to some safe haven any environmentally noxious substance. This can be accomplished by one of two ways - either by safe disposal to a "dump" where the material is rendered inert (say as an unleachable constituent in a composite substance or in an acceptable containment vault from which the substance cannot escape) - or as a constituent in a product which is to be used by society in a form which is environmentally acceptable, even though there is a cost for the use of this material rather than a profit. Generally, any disposal of material has a cost attached in either the short or the long term.

The "waste" products from steel production and the eventual obsolete scrap from steel products are increasingly being categorized as substances that have to be treated in a way which makes them harmless to our environment. A major source of "waste" products is the electric arc furnace. The quantity of dust from this operation is increasing as more steel is produced by this process. In North America EAF dust amounts to approximately 500,000 tonnes. In the U.S.A. this material has been classified by the Environmental Protection Agency (EPA) to be a hazardous substance and its safe disposal is in the process of being mandated.

2.0 NUMEROUS REVIEWS OF THE EAF DUST PROBLEM

Until 1984 the treatment of EAF dust as an industrial by-product waste was handled primarily by having this relatively low-valued material disposed in the nearest landfill.

Table I - EAF Dust Disposal

Landfill	73%
Fertilizers Use	8.9%
Zn Recovery	14.5%
Miscellaneous	3.6%

However, the potential environmental restrictions for this form of disposal had already been of concern. This was well documented in February 1982, in a study done by Lehigh University for the U.S. Department of Commerce, "Characterization, Recovery and Recycling of Electric Arc Furnace Dust".(1) In 1984 the Hazardous and Solid Waste Amendments (HSWA) were passed re the Resource Conservation Recovery Act (RCRA). The environmental status for EAF dust is well reviewed by a paper (see Apppendix I) given by Chris W. Avent and James C. Morriss III, "Environmental Regulation of Electric Furnace Dust: Alternatives and Decisions" which has been published this year (1987) in a most thorough review of the whole subject by The Iron and Steel Society.(2) Other recent reviews include the "Proceedings of the 44th Electric Furnace Conference", ISS-AIME, December 1986,(3) (see Appendix 2) and the brief summary article, (Appendix 3) in I&SM, August 1987 which reports on the meeting, "Update on EAF Dust Activities"; sponsored by the Center for Metals Production (CMP) held in Pittsburgh on June 18, 1987.

An early report by Donald R. MacRae in May 1985 reports on the CMP Project Number RP-2570-1-2, "Electric Arc Furnace Dust - Disposal, Recycle and Recovery".(4) A recent Seminar (November 19, 1987) sponsored by the American Metal Market (November 19, 1987) and the National Steel Producers Association was on electric furnace dust. The Electric Furnace Conference held December 8-11, 1987 was also once again concerned with the EAF dust problem.

The Environmental Protection Agency defined EAF dust as a solid waste and listed it as both a hazardous waste and as a sludge. The Land Ban is scheduled to become effective starting on August 8, 1988 and is to be fully in place by May 8, 1990. As there is not yet sufficient installed capacity to handle the dust it is fully expected the regulations will be delayed to come into effect at the later date only, i.e. May 8, 1990.

Despite the fact that stabilization of the hazardous metals in the dust and removal of these metals by a recovery process are both proposed to be acceptable solutions to the EAF dust problem, there is still a concern about any process of stabilization, i.e. whether or not the stabilization will hold over the long-term, is the question. This attitude gives a strong preference for treatment processes which will remove all the hazardous materials and recover them, and the iron values, from the EAF dust for reuse. This preference should be kept in mind as decisions are made and processes perfected for the treatment of the EAF dusts.

3.0 PROCESSES OF NEAR-TERM INTEREST

The reviews and articles mentioned list, review, evaluate and speculate on the technical and economic value of a great number of processes which may offer solutions to the EAF dust treatment/disposal crisis. However, there seemed to be somewhat of a concensus from those who have been contacted (see Appendix 4) that the processes of greatest near-term potential were the ones selected for discussion at the September 1987 International Lead and Zinc Study Group meeting on "Secondary Recovery of Lead and Zinc" held in Washington, D.C. These were:

- Tetronics Plasma Furnace Process
- St. Joe Flame Reactor Process
- Waelz Kiln Process
- SKF Plasmadust Process

The Tetronics process results in a concentration of metal oxides which is sufficient for the recovery of zinc metal, recovers liquid iron, and produces a non-hazardous slag.

The St. Joe process also produces a high enough concentration of metal oxides to allow zinc recovery. However, it does not recover the iron values but has a slag which is suitable to be used as a concrete aggregate.

The Waelz kiln process produces a zinc concentrate of about 55% zinc which is suitable for zinc recovery processes and an iron-bearing slag which is stated to meet with the EPA Leaching Standards.

The SKF process recovers zinc metal in a condenser and liquid iron plus a slag for disposal.

Some other processes are quite similar to these four and may become close contenders (e.g. the Elkem process and the Davy McKee process also recover zinc via the condensation of zinc vapour in a splash lead condenser).

The other approaches are either some form of direct return of the dust to a steelmaking process, (e.g. EAF dust injection trials by LTV Steel Company), or the various attempts to put the hazardous metals into a non-hazardous form (e.g. Chemfix's process to neutralize the materials by putting them into a non-leachable sludge, or Nippon's process which injects the dust in pellets into molten slag which takes up the heavy metals, for eventual safe disposal, with the zinc content collected in the baghouse).

4.0 LONG-TERM DIRECTIONS

The rush to obtain some useable solutions to the EPA requirements in the near-term will not allow new process much opportunity before the May 1990 date. However, any processes which can be perfected which will recover metal values rather than mixing them into a product which is for nonhazardous disposal should receive encouragement as potential long-term solutions to the use for EAF dust. The LB process has objectives in this direction and therefore further research should be carried out to determine what economic scope this process can offer.

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1. "Characterization, Recovery and Recycling of Electric Arc Furnace Dust", Lehigh University for U.S. Department of Commerce, February 1982.

2. "Disposal, Recycling and Recovery of Electric Furnace Exhaust Dust", Iron and Steel Society Inc., Edited by R. Kaltenhauser, 1987.

3. "Proceedings of the 44th Electric Furnace Conference", ISS-AIME, December 1986.

4. "Electric Arc Furnace Dust - Disposable, Recycle and Recovery", CMP Project RP-2570-1-2, by Donald R. MacRae, May 1985.

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LIST OF APPENDICES

I "Environmental Regulation of Electric Furnace Dust: Alternatives and Decisions"

II "Recycling Furnace Dust"

III "So Little Time, So Many Choices"

IV List of Contacts

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APPENDIX I

"Environmental Regulation of Electric Furnace Dust: Alternatives and Decisions"

by Chris W. Avent & James C. Morriss III

p 51 - 65 "Disposal, Recycling and Recovery of Electric Furnace Exhaust Dust", Iron and Steel Society Inc., 1987.

ENVIRONMENTAL REGULATION OF ELECTRIC FURNACE DUST: ALTERNATIVES AND DECISIONS

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I. INTRODUCTION

Electric arc furnace technology is utilized throughout the continental United States. Companies employing this technology generate approximately 500,000 tons of electric arc furnace ("EAF") dust annually. While the precise composition of EAF dust will vary from plant to plant, the dust characteristically contains zinc and the heavy metals, lead, cadmium, and chromium. Thus, as the United States Environmental Protection Agency ("EPA") established the guidelines for the cradle to grave regulation of hazardous waste, this large volume waste was listed as a hazardous waste. As such, EAF dust must be handled, transported, stored, and disposed in a manner consistent with the requirements of federal and state law.

In 1984, Congress amended the Resource Conservation Recovery Act by the passage of the Hazardous and Solid Waste Amendments of 1984 ("HSWA") (the amended statute is hereinafter referred to as "RCRA") 42 U.S.C.A. § 6901 et seq. (Supp. 1985). In expanding the regulatory program to be administered by EPA, Congress expressed a very strong sentiment against further land disposal of hazardous waste. The EPA has now promulgated a time schedule for systematically eliminating the land disposal of untreated hazardous waste, including EAF dust. The unmistakable purpose of HWSA was to encourage, with "hammer provisions," the waste minimization, treatment, and reclamation of hazardous waste. However, the generator of EAF dust remains subject to hazardous waste regulation even upon implementing a treatment or reclamation process.

This paper will review the present regulatory status of EAF dust, the general regulatory scheme which is currently applicable to the generation, transportation, storage, and disposal of this waste, the schedule and manner of implementation of the land disposal prohibition, the issues which must be addressed to treat or reclaim EAF dust, including the bases of liability for improper handling of the material, and other issues which the operators of electric arc furnaces should consider as they evaluate different methods of handling this waste material. The authors have emphasized issues which will confront the industry during the next several years as alternatives to land disposal are first identified, then evaluated, and finally mandated.

Obviously, electric arc furnaces are located throughout the country in many different states and local jurisdictions which may alter the basic federal scheme which underlies hazardous waste regulation in this country. Since the federal regulatory program is generally the catalyst for consistent state regulation, this article focuses on the regulations and policies of EPA. The reader should evaluate whether the peculiarities of a particular state or local hazardous waste program alter the federal approach. Federal law requires states delegated the federal program to have regulations at least as restrictive as those developed by EPA.

II. Nature of the Waste

A. The Importance of Definitions

As in almost every regulatory program, the federal regulation of hazardous waste builds upon a framework of definitions. Whether material is subject to regulation is determined by whether or not it is encompassed within a particular defined term. If the material is included, the scope of the limitations imposed upon its generation and handling may likewise be determined by whether the proposed method of handling comes within the definition of yet another term. Thus, the producer of a material must follow a stepwise process to ascertain whether, in fact, he is a generator of a solid waste, whether that solid waste is a hazardous waste, whether the proposed method of handling constitutes treatment, storage, or disposal, and whether a proposed use might be a form of reclamation which excludes the material from further regulation. The cornerstone of this definitional scheme is the term "solid waste."

B. What is a Solid Waste?

RCRA defines solid waste to mean "any garbage, refuse, sludge from a waste water treatment plant, water supply treatment plant, or air pollution control facility and other discarded material, including solid, liquid, semisolid, or contained gaseous material resulting from industrial, commercial, mining, and agricultural operations, and from community activities." 42 U.S.C.A. § 6903(27) (1983). Thus, from the basic statutory definition, EAF dust would appear to be a "solid waste" if discarded. In fact, EPA has regulated the material in a much broader manner. Solid waste is defined to include, not only discarded materials, but certain inherently waste-like materials and, in specific instances, materials which are recycled. 40 C.F.R. § 261.2(a) (1985). The term "solid waste" is significant in that if a material is not a "solid waste," it is not a hazardous waste and, therefore, not subject to the comprehensive regulatory requirements imposed by RCRA. EAF dust has not escaped the broad interpretation of the term "solid waste."

C. The Listing of EAF Dust

Section 3001 of RCRA directs EPA to identify the characteristics of hazardous waste and to list particular hazardous wastes. 42 U.S.C.A. § 6921 (Supp. 1985). Given the large volume of EAF dust generated, the waste received individual attention from EPA and was listed as Hazardous Waste No. K061, "Emission Control Dust/Sludge from the Primary Production of Steel in Electric Furnaces." 40 C.F.R. § 261.32 (1985). EPA concluded that EAF dust "may pose a present or potential hazard to human health and the environment when improperly transported, treated, stored, disposed of, or otherwise managed." Listing Background Document: Electric Furnace Production of Steel, EPA. EPA's decision to list EAF dust as a hazard waste was based upon the following factors:

> 1. The emission control dusts/sludges contain significant concentrations of the toxic metals chromium, lead, and cadmium.
>
> 2. Lead, chromium and cadmium have been shown to leach in harmful concentrations from waste samples subjected to both a distilled water extraction procedure and the extraction procedure described in . . . [the then proposed regulations].
>
> 3. A large quantity of these wastes (a combined total of approximately 500,000 metric tons) is generated annually and is available for disposal. There is thus a likelihood of large scale contamination of the environment if these wastes are mismanaged.
>
> 4. The wastes typically are disposed of by being dumped in the open, either on-site or off-site, thus posing a realistic possibility of migration of lead, cadmium, and chromium to underground drinking water sources. These metals persist virtually indefinitely, presenting the serious threat of long-term contamination.
>
> 5. Off-site disposal of these wastes will increase the risk of mismanagement during transport.

Id.

EAF dust is also encompassed in the rather misleading term "sludge." A sludge includes, among other things, any solid, semisolid, or liquid waste generated from an industrial air pollution control facility. 40 C.F.R. § 260.10 (1985).

D. Implications of Being a Listed Sludge

As a sludge listed in 40 C.F.R. § 261.32, EAF dust remains subject to regulation as a hazardous waste even when it is destined for reclamation. Id. § 261.2(c)(3). Further, the EPA has established the "derived from" rule and the "mixture rule" which taint materials derived from or mixed with a listed hazardous waste. The "derived from" rule creates a presumption that any solid waste generated from the treatment, storage, or disposal of a hazardous waste, including any sludge, spill residue, ash, emission control dust, or leachate is a hazardous waste. Id. § 261.3(c). In the case of a waste that is hazardous merely by virtue of a characteristic identified in Subpart C of 40 C.F.R. Part 261 ("Subpart C"), this presumption is rebuttable simply by demonstrating that the waste derived from the hazardous waste no longer exhibits a hazardous waste characteristic. However, in contrast, the regulatory stigma of listing associated with EAF dust can only be removed from a waste derived from the treatment, storage, or disposal of EAF dust by a formal delisting petition. Id. § 261.3(c). Similarly, the "mixture rule" creates a presumption that a solid waste is a hazardous waste if it a mixture of a solid waste and a hazardous waste. Once again, the presumption is only rebuttable in the case of a waste that is hazardous merely by virtue of a characteristic identified in Subpart C. In the case of a listed hazardous waste, formal delisting is required. Id. § 261.3(a)(iv).

Thus, by defining EAF dust as a listed hazardous waste and as a sludge, EPA has established the substantive regulatory scheme which applies to this material, to materials derived from it, or mixtures containing it, from cradle to grave. To alter this regulatory scheme, the generator must either reclaim the dust in such a way as to produce a useful product or treat the material in a manner which permits the generator to pursue a delisting petition.

III. PRESENT REGULATION OF EAF DUST

A. General

RCRA was enacted as a cradle to gravel solution to hazardous waste generation. The statute and the regulations promulgated by EPA regulate generation, transport, storage, and disposal of EAF dust. Further, even the reclamation of dust is subject to regulation.

B. Standards Applicable to Generators

EPA has defined the term "generator" to include any person whose act or process produces a hazardous waste. Id. § 260.10. Thus, a steel maker producing EAF dust is a "generator" and must accordingly comply with the generator obligations set forth in 40 C.F.R. Part 262 (1985). The initial obligation of a generator of a solid waste is to ascertain whether that solid waste is a hazardous waste. Id. § 262.11. This is an easy task for the steel maker since EAF dust is specifically listed as such. As the "cradle" in the cradle to grave scenario, the generator must comply with requirements designed to initiate the hazardous waste tracking system. The initial step in this process is obtaining an EPA identification number before treating, storing, disposing of, transporting, or offering to transport hazardous waste. Id. § 262.12. A second safeguard in the waste tracking system is the prohibition which forbids a generator to offer his hazardous waste to transporters or to treatment, storage or disposal facilities which have not received an EPA identification number. Id. The regulations then proceed to impose upon the generator an obligation to manifest any hazardous waste shipment. This shipping ticket is designed to insure that the wastes arrive at a designated consignee and if they fail to arrive, that both the generator and the agency are made aware of that fact. Id. §§ 262.20 - 262.23. The regulations also impose upon the generator an obligation to maintain records including copies of manifests for shipments off-site. Every two years, a generator must prepare a report advising EPA (or authorized state agency) of its waste handling practices. Id. § 262.41.

C. Standards Applicable to Transporter

The transporter is the second link in the cradle to grave management of hazardous waste. Obviously, the manifest system is embodied in requirements applicable to transporters. A transporter may not accept hazardous waste from a generator unless it is accompanied by a manifest signed by the generator in accordance with the generator requirements. The transporter must then insure that the manifest accompanies the hazardous waste and that the waste is delivered consistent with the instructions contained in the manifest. The transporter must also maintain a copy of the manifest for himself. Regulations also impose upon the transporter an obligation to take appropriate and immediate action in the event of discharge of hazardous waste and to provide notification to the National Response Center in such an event. Id. Part 263.

D. Standards Applicable to Owners and Operators of Treatment, Storage, and Disposal Facilities

Most of the EAF dust being generated in the United States is being landfilled in "monofills" at or in the immediate vicinity of the plant where the dust is generated. A monofill is a landfill which receives a single waste stream which is homogeneous in nature. Regulations prescribe the manner in which this waste will be handled, stored, and disposed. While Congress has directed EPA to terminate land disposal of untreated waste, including EAF dust, the importance of the standards applicable to owners and operators of Treatment, Storage and Disposal ("TSD") facilities transcends current management practices. (The forthcoming "Land Ban" is described in detail in Section IV below.) Even if EAF dust is treated or reclaimed, the storage of dust prior to such processing and possibly the handling and storage of certain of the products of processing remain subject to regulatory requirements. Further, if such processing results in the generation of a waste material or is merely a predisposal treatment technology, absent the delisting of the waste material, the handling and disposal of the waste would remain subject to hazardous waste regulations. Id. § 261.3(c).

The following brief summary will illustrate the comprehensive nature of the existing regulatory framework:

1. EPA Identification Number

The final step in the cradle to grave process is to insure that the owner/operator of the TSD facility is properly identified. Thus, the owner/operator of the TSD facility must obtain an EPA identification number. Id. § 264.11.

2. Waste Analyses

The regulations require that an owner/operator obtain a detailed chemical and physical analysis of a representative sample of a waste before treating, storing or disposing of that waste. Given the relatively homogeneous nature of EAF dust, a one time analysis may well be sufficient to comply with the requirement that the analysis be repeated as necessary to insure that it is accurate and up-to-date. Of course, if the nature of the charge to the electric arc furnace should change significantly, with the potential for affecting the character of the waste, the owner/operator must perform additional analyses. Id. § 264.13.

3. Security

The regulations require the owner/operator to prevent the unknowing entry and minimize the possibility of unauthorized entry of persons or livestock into the active portion of the TSD facility. Absent a demonstration that physical contact with the waste, structures, or equipment will not injure persons or livestock or that disturbance of the waste by persons or livestock will not cause a violation of the requirements applicable to owners/operators, the owner/operator must have either 24 hours surveillance or an artificial (e.g., fence) or natural barrier (e.g., cliff) which completely surrounds the active portion of the facility and a means to control entry. Signage that warns against unauthorized entry is required. Id. § 264.14.

4. General Inspection Requirements

The owner/operator must develop and follow a written schedule for inspecting the facility for malfunctions, deterioration, operator errors, and discharges which may cause the release of hazardous waste constituents to the environment or pose a threat to human health. The owner/operator is also required to remedy any deterioration or malfunction detected during the inspection and maintain a log of the inspections and corrective action taken. Id. § 264.15.

5. Personnel Training

Regulations require that facility personnel must complete a program of classroom instruction and on-the-job training which teaches them to perform their duties in a manner that insures the facility's compliance with the requirements of this part. The program must be directed by the person trained in hazardous waste management procedures. In addition to completing an initial program within six months after the date of their employment or assignment to the facility, the personnel must take part in an annual review of the initial training. The operator must maintain a written job description for those involved in hazardous waste management practices and a written description of the type and amount of training provided as well as records documenting the training. Id. § 264.16.

6. Preparedness and Prevention

The regulations require that facilities be designed, constructed, maintained, and operated to minimize the possibility of fire, explosion, and any unplanned sudden or nonsudden release of hazardous waste or hazardous waste constituents. Absent a demonstration that the hazards posed by a particular waste could not require a particular kind of equipment, facilities must have such items as internal communication systems or alarms, a method of summoning emergency aid, portable fire extinguishers, and water at adequate volume and pressure to supply water hoses or other firefighting equipment. Further, the owner/operator must attempt to make arrangements with local authorities whereby police, fire departments and emergency response teams are made familiar with the layout and practices employed at the facility. Further, local hospitals are to be familiarized with the properties of the waste handling facility to better enabled them to respond to emergency needs. The steel maker should assess whether any of these procedures legitimately apply to the handling and disposal of EAF dust. Id. §§ 264.30 - 264.37.

7. Contingency Plan and Emergency Procedures

Each owner/operator must have a contingency plan designed to minimize the hazards to human health or the environment from unplanned events such as fires, explosions, or any sudden or nonsudden release of hazardous waste. The plan must describe the actions of facility personnel in such event, describe arrangements made with local and emergency authorities, and list the names of emergency coordinators who will operate to implement the plan in the event of an emergency condition. Finally, the plan must include a list of emergency equipment, describe emergency procedures, and include an evacuation plan for facility employees.

A steel manufacturer is faced with the dilemma of what emergency involving EAF dust could justify the implementation of an emergency contingency plan. Perhaps a major spill, baghouse fire, or similar incident involving the release of a large volume of the material in an uncontrolled manner would qualify for implementation of a contingency plan. Id. §§ 264.50 - 264.56.

8. Manifest System

As reviewed above, the manifest system is the mechanism by which EPA insures that wastes reach their final "grave." Accordingly, the owner/operator of a TSD facility must close the loop by completing the manifest, returning a copy of the manifest to the generator and retaining a copy for his files. In addition to recordkeeping associated with manifest requirements, the owner/operator must maintain an "operating record" which includes a description of the hazardous waste received, location of the hazardous waste within the facility, records of waste analyses, reports of incidents involving the implementation of a contingency plan, inspection reports, groundwater monitoring results and closure and postclosure cost estimates. While this operating record need not be compiled in one volume, it will likely be the first "record" requested during an on-site inspection by agency personnel. Therefore, all of the materials making up the operating record should be maintained in an accessible place available for inspection. Id. §§ 264.70 - 264.77.

9. Groundwater Monitoring

The regulations prescribe detailed procedures for the installation and maintenance of a groundwater monitoring network at hazardous waste management units. The purpose of the groundwater monitoring requirements is detection, evaluation, and correction of the effects of a hazardous waste management unit on groundwater. Id. §§ 264.90 - 264.100.

10. Closure and Postclosure

The owner/operator must develop a plan for the closure of the facility and depending upon the manner of closure, for the provision of postclosure care for 30 years following the date of closure. The closure plan must detail the steps to be taken to decontaminate the facility or provide for long term maintenance of waste left in place. Id. §§ 264.110 - 264.120.

11. Financial Requirements

In order to insure that adequate funds are available to accomplish closure and postclosure care, the owner or operator must comply with financial assurance requirements. The owner or operator may choose from a number of financial mechanisms including establishing a trust fund, acquiring a bond, letter of credit, or closure insurance, or demonstrating annually that the company meets a financial test predicated upon the financial strength of the company or its corporate parent. Id. §§ 264.140 - 264.145.

12. Liability Requirements

Perhaps one of the most frustrating provisions contained in the regulations applicable to an owner/operator of a TSD facility is the liability requirement. Id. § 264.147. This provision requires that an owner/operator maintain sudden accidental insurance coverage of one million dollars per occurrence with an annual aggregate of two million dollars and nonsudden accidental insurance of three million dollars per occurrence with an annual aggregate of six million dollars. In both cases, the coverage must be exclusive of legal defense costs.

The mounting costs of addressing environmental claims as well as hardships encountered by insurance carriers in other liability areas has all but eliminated insurance coverage at these levels. Where insurance can be obtained, the cost of such policies is prohibitive and the coverage is often inadequate to satisfy the statutory minimums. The only option to liability coverage is the satisfaction of a financial test predicated upon the financial strength of the company or its corporate parent. Id. § 264.147. Of course, many owners and operators of TSD facilities were unable to satisfy this financial test. HSWA, the 1984 amendments to RCRA, terminated the interim status of any facility that could not satisfy these liability requirements as of November 8, 1985. Despite the argument that management of certain wastes, including EAF dust, poses little risk to the environment, absent a variance reducing the level of required insurance to a level obtainable, companies may well be forced to terminate proper on-site practices for more costly and possibly less secure off-site waste management.

IV. LAND BAN

A. Section 3004 of RCRA

1. General

HSWA established a strong statutory presumption against land disposal by prohibiting the continued land disposal of hazardous wastes beyond specified dates unless such disposal is determined to be protective of human health and the environment 51 Fed. Reg. 19300 (1986). The statutory basis for a limitation on the land disposal of EAF dust is contained in section 3004(g) of RCRA. 42 U.S.C.A. § 6924(g) (Supp. 1985). This section imposed upon EPA an obligation to submit to Congress by November 8, 1986, a schedule for reviewing all hazardous waste listed under RCRA unless otherwise specifically scheduled by the statute and for taking action with respect to each waste. EPA was directed to base the schedule upon a ranking of listed hazardous waste developed from the intrinsic hazard and the volume of the wastes such that decisions regarding the land disposal of high volume hazardous wastes with high intrinsic hazard would be scheduled first and low volume waste with lower intrinsic hazard would be scheduled last. Section 3004(g)(4) of RCRA requires the EPA to make determinations on land disposal within the following time frames:

a. At least one-third (1/3) of all ranked and listed hazardous waste by August 8, 1988;

b. At least two-thirds (2/3) of all ranked and listed hazardous waste by June 8, 1989; and

c. All remaining ranked and listed hazardous waste and all hazardous waste identified by characteristic under § 3001 of RCRA by May 8, 1990.

Id.

Section 3004(g)(5) of RCRA directs EPA to promulgate final regulations prohibiting one or more methods of land disposal except for methods which the Administrator determines will be protective of human health and the environment for as long as the waste remains hazardous. This determination by the Administrator is to be made considering:

a. The long term uncertainties associated with land disposal;

b. The goal of managing hazardous waste in the appropriate manner in the first instance; and

c. The persistence, toxicity, mobility, and propensity to bioaccumulate of such hazardous waste and their hazardous constituents.

A method of land disposal may not be determined to be protective of human health and the environment unless upon application by an interested person, it has been demonstrated to the Administrator to a reasonable degree of certainty that there will be no migration of hazardous waste constituents from the disposal unit or injection zone for as long as the wastes remain hazardous. Id. § 6924(g)(5).

2. Availability of a Variance

The Administrator may establish an effective date different from the effective date which would otherwise apply to a waste under Section 3004(g) of RCRA. Such action by the Administrator must establish an effective date on the basis of the earliest date on which adequate alternative treatment, recovery, and disposal capacity which protects human health and the environment will be available. RCRA provides that in no event, shall the Administrator establish another effective date later than two years after the effective date of the prohibition which would otherwise apply. Id. § 6924(h)(2).

In addition to independent action by the Administrator, the agency can act on a case-by-case basis to grant an extension for up to one year where an applicant demonstrates the existence of a binding contractual commitment to construct or otherwise provide such alternative capacity and that due to circumstances beyond the control of the applicant that alternative capacity cannot be reasonably be made available by the effective date. This extension can be renewed for one additional year. Further, during the period of the extension, the hazardous waste may be disposed of in a landfill only if that facility meets the minimum technology standards (double liner, leachate collection, and leak detection) contained in Section 3004(o). Id. § 6924(h)(3) - (4).

3. Treatment Standards

In addition to prohibiting land disposal of wastes, EPA is to promulgate regulations specifying those levels or methods of treatment, if any, which substantially diminish the toxicity of the waste or substantially reduce the likelihood of migration of hazardous constituents from the waste so that the short term and long term threats to human health and the environment are minimized. If a hazardous waste has been treated to a level or by a method specified by EPA in this manner, that waste or residue is not subject to the prohibition on land disposal. Id. § 6924(m).

4. Failure to Achieve Statutory Deadlines

The regulations also prescribe some relief in the event that EPA fails to meet the statutory deadlines contained in the Act. If, by the statutory deadline for any hazardous waste in the first one-third (1/3) or second one-third (1/3) of the schedule, the EPA fails to promulgate regulations establishing conditions under which the hazardous wastes may be land disposed, the hazardous wastes may be disposed in a landfill or surface impoundment if the facility is in compliance with the minimum technology requirements set forth in section 3004(o) of RCRA. Id. § 6924(o). Further, prior to the disposal, the generator must certify to EPA that the generator has investigated the availability of treatment capacity and has determined that the use of the landfill or surface impoundment is the only practical alternative to treatment currently available to the generator. Id. § 6924(g)(6)(A) -(B).

If the EPA fails to set treatment standards for any of the scheduled listed wastes by May 8, 1990, the wastes are prohibited from land disposal as of that date unless the waste is the subject of a successful petition pursuant to section 3004(g)(5) of RCRA or the EPA establishes a later deadline pursuant to section 3004(h). 42 U.S.C.A. § 6924(g)(6)(C) (Supp. 1985). Interestingly, EPA has interpreted the statute as allowing more flexibility as to wastes identified as hazardous by characteristic ("identified wastes"). The agency has announced that the automatic prohibition applicable to listed wastes does not apply to identified wastes.

5. Schedule for EAF Dust

In response to the congressional directive to schedule action prohibiting land disposal EPA issued a final rule establishing a schedule for land disposal restrictions on May 28, 1986. 51 Fed. Reg. 19,300 (1986) (to be codified at 40 C.F.R. §§ 268.10 - .13). EPA, noting section 3004(g)(2) which require the schedule to be based on a ranking of wastes such that high volume wastes with a high intrinsic hazard are ranked first and low volume wastes with a lower intrinsic hazard ranked last, adopted a method for ranking the wastes by multiplying a "toxicity score" for each waste and a "volume score" for waste. Id. at 19,302. The schedule applied to all the waste that were listed as hazardous in 40 C.F.R. Part 261 as of November 8, 1984, except for the spent solvent wastes listed as

EPA hazardous wastes Nos. F001, F002, F003, F004 and F005 and dioxin-containing hazardous wastes Nos. F020, F021, F022, F023, F026, F027 and F028. These two categories of wastes were excluded from the schedule because RCRA requires decisions on them to be made within twenty-four (24) months of enactment. Id. at 19,304.

Based upon the "toxicity score" multiplied by the "volume score," the EPA ranked the various wastes listed by 40 C.F.R., Part 261 and then divided the list into groups of thirds. For the first group, the EPA will take action under section 3004(g)(5) (to prohibit land disposal for all methods not found to be protective of human health and the environment) and section 3004(m) (to specify levels or methods of treatment), by August 8, 1988. Id. at 19,305. EAF dust is included in this first group of wastes.

B. Regulatory Approach To Implementing the Land Ban

1. Proposed Approach

On January 14, 1986, EPA proposed a framework by which treatment standards for hazardous waste restricted from land disposal would be established. 51 Fed. Reg. 1602 et seq. (1986). This approach utilized technology based levels of treatment in conjunction with risk based standards called "screening levels." The technology based levels were derived from the performance of Best Demonstrated Available Technology ("BDAT"). In contrast, the screening levels specified maximum concentration levels of individual hazardous constituents in extracts of hazardous wastes. Under the proposed rules, if the application of BDAT treatment resulted in concentration levels equal to or lower than the proposed screening levels, the agency would issue the screening level as a treatment standard, capping off required BDAT treatment. If, however, the application of BDAT treatment resulted in concentration levels less stringent than the screening levels, but the treatment still realized substantial reductions in toxicity or mobility and did not propose greater risk than land disposal, then the technology based level would become the treatment standard and the screening level would remain as a goal that could reached as new technologies emerge. Id. and 51 Fed. Reg. 40,575 (1986). In the alternative, EPA proposed an approach based entirely upon technology based standards expressed as BDAT. Id.

EPA received a number of comments in opposition to the "screening level" approach including comments from eleven members of Congress. The congressional representatives argued that the approach did not fulfill the intent of Congress and that Congress intended to direct the agency to set treatment standards based on the capabilities of existing technologies. While not acquiescencing to the position that the agency lacked the authority to set risk based screening levels, EPA on November 7, 1986, promulgated its final framework for development of treatment standards based exclusively on levels achieved by BDAT.

2. Final Regulatory Framework

On November 7, 1986, EPA promulgated its final regulatory framework implementing land disposal restrictions and promulgating treatment standards. 51 Fed. Reg. 40,572 et seq. (1986). In addition to specifying the approach to establishing BDAT treatment standards, the regulations address petitions demonstrating land disposal to be protective of human health and the environment, variances from treatment standards, and applications for case-by-case extensions. Further, the agency describes those instances in which it will act to grant a national variance of the land disposal prohibition authorized by RCRA. Id.

3. Procedure for Determination of Best Demonstrated Available Treatment Technology (BDAT)

a. Waste Treatment Groups and "Demonstrated" Technologies

The process of establishing BDAT begins with the identification of waste treatability groups. These groups include wastes with similar physical and chemical properties in order to account for differences in types of treatment used and the effectiveness of treatment on different wastes. After this subcategorization of the RCRA Hazardous Waste Codes has been completed, EPA will identify "demonstrated" treatment technologies. EPA will not base demonstrated treatment technologies upon emerging or innovative technologies, for EPA has interpreted the statute as requiring that the technology actually be available at a full scale facility in operation for the waste or similar wastes. In the absence of a full scale treatment operation for the waste or a waste having similar treatability characteristics, the agency will be unable to identify any "demonstrated" treatment technologies. Id. at 40,588.

b. Determination of Availability

After identifying the "demonstrated technologies", EPA eliminates any of these candidates which are deemed "not available." The criteria for this determination of availability as as follows:

(1) The technology does not present a greater total risk than land disposal;

(2) If the technology is a proprietary or

patented process, it can be purchased from the proprietor;

(3) The technology provides for substantial treatment; and

(4) The technology is not prohibited under Sections 3004(n) because of air emissions.

Id. at 40,589.

Of these four criteria, the first, regarding the comparative risk assessment and third, regarding the definition of substantial treatment, merit further comment. EPA will evaluate the risk associated with treatment technologies and land disposal methods. Any technologies that are found to present a greater total risk than land disposal of the untreated waste will be excluded as being unavailable. In order to be available the demonstrated technology must also result in substantial treatment of the waste. Id. Section 3004(m) of RCRA specifies that the treatment technology must "substantially diminish the toxicity" of the waste or "substantially reduce the likelihood of migration of a hazardous constituent." 42 U.S.C.A. § 6924(m) (Supp. 1985). EPA will consider the following factors on a case by case basis in an effort to evaluate whether or not the technology is substantial:

(1) The number and types of constituents treated;

(2) Performance (concentration of constituents and the treatment residuals); and

(3) Percentage of constituents removed.

51 Fed. Reg. 40,589 (1986).

c. Collection and Evaluation of Performance Data

After identifying available demonstrated technologies, EPA will ascertain whether treatment performance data are available from full scale technologies, pilot scale technologies, or from bench scale technologies. If no such data is available, EPA, or an applicant seeking review and approval of its proposed treatment technology, will perform full scale, pilot, or bench scale treatment on waste samples using the demonstrated available treatment technologies. EPA must evaluate whether the performance data obtained is representative of a well designed and operated system. Ideally, for all treatment data EPA will have associated design and operating data. However, in the absence of such data, EPA will use engineering judgment based on a comparison of constituent concentration before and after treatment to determine whether the data reflect a well designed and operated treatment system. The Toxicity Characteristic Leaching Procedure ("TCLP") and a total waste analysis for constituents of concern will then be performed on the treatment residuals. The data from these analyses will be evaluated by the agency in order to determine whether the technology substantially diminished the toxicity and/or mobility of the waste. At this point the agency is in a position to specify BDAT treatment standards based upon the demonstrated technology for various waste treatability groups.

4. Determination of Land Ban Effective Dates

HSWA authorizes EPA to grant a nationwide variance from the statutory effective date for up to two years if adequate alternative treatment, recovery, or disposal capacity protective of human health and the environment is not available. Under the final framework announced on November 7, 1986, EPA will develop estimates of treatment capacity needed versus the capacity available in assessing whether such extension should be granted. The determinations of capacity will be made upon a national capacity basis. Since EPA will develop treatment standards for waste groups, the determination of the quantity of waste requiring specific treatment, and the available capacity will also be based upon waste groups. In evaluating available capacity EPA will consider on-line facilities and planned facilities and capacity extensions that will be on-line by the effective date of the land prohibition. Planned facilities are facilities that are under development or under construction. In response to comments concerning the inclusion of planned facilities in the capacity estimate, EPA indicated that it will consider planned capacity only if it is reasonably certain the facility will be on-line by the effective date. In defining available capacity or surplus capacity EPA may include any of the following:

a. Commercially available capacity;

b. Private capacity which can be used to process additional waste produced by the facility; and

c. Private capacity where the owner is willing and able to accept waste from other generators, i.e. to provide commercial services.

Id. a

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Id. at 40,599-40,601.

V. DELISTING PETITIONS, SECTION 3004(g)(5) PETITIONS, AND APPLICATIONS FOR EXTENSIONS AND VARIANCES

A. General

Several different administrative procedures may become increasingly important to generators of EAF dust. The first is the delisting petition filed pursuant to 40 C.F.R. §§ 260.20 and 260.22. The remaining procedures apply to various aspects of the future prohibition of land disposal of EAF dust and include the section 3004(g)(5) petition demonstrating land disposal to be protective of human health and the environment, the application for a case-by-case extension of the deadline for land disposal and the variance from the treatment standard adopted by EPA pursuant to section 3004(m).

B. Delisting Petitions

1. The Significance of Delisting Petitions

As listed hazardous wastes, EAF dust, any waste derived from the treatment, storage, or disposal or EAF dust, and mixtures of EAF dust and solid waste remain subject to regulation as hazardous waste unless specifically delisted pursuant to 40 C.F.R. §§ 260.20 and 260.22. Following delisting, the material is no longer a hazardous waste unless it is hazardous by virtue of one of the characteristics of hazardous waste. The generator need only comply with state regulations applicable to nonhazardous solid waste. Typically, such procedures are substantially less burdensome.

2. General Procedures

Any person may petition the Administrator to modify or revoke any provision in the federal hazardous waste regulatory scheme in 40 C.F.R. Parts 260 through 265. 40 C.F.R. § 260.20 (1985). 42 U.S.C. § 6974 (1983). Petitions must be submitted to the Administrator of EPA by certified mail and must include the following:

a. The petitioner's name and address;

b. A statement of the petitioner's interest in the proposed action;

c. A description of the proposed action, including (where appropriate) suggested regulatory language; and

d. A statement of the need and justification for the proposed action including any supporting tests, studies, and other information.

3. Delisting for EAF Dust

EAF dust is listed as a toxic "T" waste under 40 C.F.R. § 261.32. In order to demonstrate that EAF dust or materials derived from its treatment, storage, or disposal should be delisted, the petitioner must satisfy the requirements in 40 C.F.R. § 60.22. First, the petitioner must demonstrate that the waste does not meet any of the criteria under which the waste was listed as a hazardous waste. Further, based upon the complete application the Administrator must determine, where he has a reasonable basis to believe that factors other than those for which the waste was listed could cause the waste to be a hazardous waste, that such factors do not warrant retaining the waste as a hazardous waste. 50 Fed. Reg. 28,742 (1985) (to be codified at 40 C.F.R. § 260.22). The petitioner must show that the waste does not exhibit any of the characteristics defined in sections 261.21, 261.22, 261.23, or 261.24. Finally, the petitioner must demonstrate that the waste does not contain the constituent or constituents that caused the Administrator to list the waste or although containing one or more of the hazardous constituents giving rise to the listing is not capable of posing a substantial present or potential hazard to human health or the environment when *improperly* treating, stored, transported or disposed of or otherwise managed. 40 C.F.R. § 261.11(a)(3). In making this determination, the Administrator is directed to consider the following factors:

a. The nature of the toxicity presented by the constituent;

b. The concentration of the constituent in the waste;

c. The potential of the constituent or any toxic degradation product of the constituent to migrate from the waste into the environment under the types of improper management considered;

d. The persistence of the constituent or any toxic degradation product of the constituent;

e. The potential for the constituent or any toxic degradation product of the constituent to degrade into nonharmful constituents and the rate of degradation;

f. The degree to which the constituent or any degradation product of the constituent bioaccumulates in the ecosystems;

g. The plausible type of improper management to which the waste could be subjected;

h. The quantities of the wastes generated at individual generation sites on a regional or national basis;

i. The nature and severity of the human health and environmental damage that has occurred as a result of the improper management of the waste containing the constituent;

j. Actions taken by other governmental agencies or regulatory programs based on the health or environmental hazard posed by the waste or waste constituent;

k. Such other factor as may be appropriate.

4. Contents Of The Delisting Petition

In addition to the information required by 40 C.R.F. § 260.20, the following information must be included in the application:

a. The name and address of the laboratory facility performing the sampling or tests of the waste;

b. The names and qualifications of the persons sampling and testing the waste;

c. The dates of the sampling and testing;

d. The location of the generating facility;

e. A description of the manufacturing processes or other operations and feed materials producing the waste and an assessment of whether such processes, operations, or feed material can or might produce a waste that is not covered by the demonstration;

f. A description of the waste and an estimate of the average and maximum monthly and annual quantities of the waste covered by the demonstration;

g. Pertinent data on and discussion of the factor delineated in the respective criterion for listing a hazardous waste;

h. A description of the methodologies and equipment used to obtain the representative samples by a description of the sample handling and preparation techniques including techniques used for extraction, containerization and preservation of samples;

i. A description of the test performed;

j. The names and model numbers of instruments used in performing the test and a certification by an authorized representative of the accuracy and authenticity of the information submitted.

5. Guidance Manual For Delisting Petitions

EPA has published a guidance manual for delisting petitions. This document was published in April of 1985 and is identified as Document No PB85-194488. The document is available through the U.S. Department of Commerce National Technical Information Service.

C. Section 3004(g)(5) Petitions Demonstrating Land Disposal to be Protective of Human Health and the Environment

1. General

Section 3004(g)(5) of RCRA authorizes the Administrator to determine that a land disposal method is protective of human health and the environment if an applicant can demonstrate to a reasonable degree of certainty that there will be no migration of hazardous constituents from the disposal unit or injection zone for as a long as the wastes remain hazardous. 42 U.S.C.A. § 6924(g)(5) (Supp. 1985). EPA has identified three scenarios that may satisfy the requirements of the statutory standard of "no migration". They are as follows:

a. Where environmental parameters are such that no detectable migration of hazardous constituents would occur from the disposal unit (*e.g.*, a monofill located in an arid climate with no groundwater recharge);

b. Where an active chemical or physical process such as neutralization of corrosive wastes insures that no hazardous wastes remain in a unit; and

c. Temporary storage of hazardous waste in a land based unit such as an indoor waste pile where engineered containment systems are effective over the period the waste remains in storage.

51 Fed. Reg. 40,605 (1986).

2. Contents of Petition

Regulations require that a person seeking such an exemptions from the prohibition of land disposal submit a petition to the Administrator demonstrating "to a reasonable degree of certainty, that there will be no migration of hazardous constituents from the disposal unit or injection zone for as long as the waste remain hazardous." 51 Fed. Reg. 40,640 (1986) (to be codified at 40 C.F.R. § 268.6). The demonstration must also include the following components:

a. An identification of the specific waste and the specific unit for which the demonstration will be made;

b. A waste analysis to describe fully the chemical and physical characteristics of the subject waste; and

c. A comprehensive characterization of the disposal unit site including an analysis of background air, soil, and water quality.

Id.

3. Evaluation Criteria

The demonstration contained in the petition must meet the following criteria.:

a. All waste and environmental sampling, test, and analyses data must be accurate and reproducible to the extent that state of the art techniques allow;

b. All sampling, testing, and estimation techniques for chemical and physical properties for waste and all environmental parameters must have been approved by the Administrator;

c. Simulation models must be calibrated for specific waste and site conditions, and verified for accuracy by comparison with actual measurements;

d. A quality assurance and quality control plan that addresses all aspects of the demonstration must be approved by the Administrator; and

e. An analysis must be performed to identify and quantify any aspects of the demonstration that contribute significantly to uncertainty.

Obviously, EPA has created a significant burden for the petitioner to overcome. EPA indicated that the "no migration" standard would be violated in a situation where unacceptable concentrations of hazardous constituents are occurring at the waste management boundary, even though the concentration at a potential receptor site some distance from the waste management boundary is below an acceptable health based standard. 51 Fed. Reg. 40,605 (1986).

D. Case-by-Case Extensions of the Effective Date

Section 3004(h)(3) provides for a case by case extension from the deadline for prohibition of land disposal for a person who generates or manages a restricted hazardous waste who can demonstrate that due to circumstances beyond his control alternative capacity reasonably cannot be made available by the applicable effective date and that he has entered into a binding contract to construct or otherwise provide for such alternative treatment recovery or disposal capacity. 42 U.S.C.A. § 6924(h)(3). The final regulations require that the applicant make a good faith effort to locate and contract with alternative technologies nationwide. EPA observed that the case-by-case extension process was intended to cover only those rare situations when an individual applicant can demonstrate the capacity will not be available to him even if national capacity is otherwise sufficient. 51 Fed. Reg. 40,604 (1986). Thus, EPA believes that the requirement of a good faith effort to locate available capacity is consistent with the legislative intent. In assessing whether the lack of capacity is beyond the applicant's control, EPA will consider the feasibility of providing alternative capacity during the period of the requested extension. The determination of feasibility may involve consideration of the technical and practical difficulties associated with providing alternative capacity. The agency will require that the applicant demonstrate that after the extension sufficient capacity will exist for the waste covered by the application. The applicant must provide a detailed schedule for obtaining the required operating and construction permits or an outline of how and when alternative capacity will be available. The applicant must also demonstrate that he has arranged for adequate capacity to manage his waste during the extension and must document the location of all sites in which the waste will be managed. Finally, any waste managed in a surface impoundment or landfill during the extension period must be handled consistent with minimum technology requirements established by section 3004(o). 51 Fed. Reg. 40,639 (1986) (to be codified at 40 C.F.R. § 268.5).

The Administrator, following notice and opportunity for comment and after consultation with the appropriate state agencies, may grant an extension of up to one year from the effective date of the land disposal ban. The Administrator may extend the extension for up to one additional year upon request of the applicant if the demonstration required in the original application can still be made.

E. Treatability Variance

1. General

In the final regulations, the agency recognized that there may be a unique waste that cannot be treated to the level specified as the treatment standard. The regulations authorize generators or owner/operators to submit a petition to the Administrator requesting a variance from the treatment standard. On September 5, 1986, EPA published a notice in the Federal Register indicating that any such variance would be processed as a rule making proceeding pursuant to the agency's authority existing under section 7004(a) of RCRA. 51 Fed. Reg. 31,783-31,789 (1986). In response to comments on that notice, the agency has in the final rules adopted procedures for such variances. Given the time available to the agency to consider an appropriate procedure, the initial procedures will be similar to those used in rule making petitions under 40 C.F.R. § 260.20. Id. at 40,605. However, EPA is seeking additional comments regarding procedures for processing such variances and may consider establishing different and more streamline procedures in the future. Id. at 40,606.

2. Burden of Proof

The variance petitions must demonstrate that the treatment standard established for a given waste cannot be met. The demonstration can be made by showing that attempts to treat the waste by available technologies are not successful or through appropriate analyses of the waste which demonstrate the waste cannot be treated to the specified levels. Id.

3. Contents of the Petition

Each petition must be submitted in accordance with the procedures of § 260.20 and should contain the following information:

a. The petitioner's name and address;

b. The statement of the petitioner's interest in the proposed action;

c. Name, address, and EPA Identification number of the facility generating the waste and the name and telephone number of the plant contact;

d. The processes and feed materials generating the waste and an assessment of whether such processes or feed materials may produce a waste that is not covered by the demonstration;

e. A demonstration of the waste sufficient for comparison with the waste considered by the agency in developing BDAT and an example of the average and maximum monthly and annual quantities of waste covered by the demonstration;

f. If the waste has been treated, a description of the system used for treating the waste, including the process design, operating conditions and an explanation of the reasons the treatment standards are not achievable or are based on inappropriate technology for treating the waste;

g. A description of the alternative treatment systems examined by the petitioner, a description of the treatment system deemed appropriate by the petitioner for the waste in question, and as appropriate the concentrations of the treatment residual or extract of the treatment residual that can be achieved by applying such treatment to the waste;

h. The dates of the sampling and testing;

i. A description of the methodologies and equipment used in obtaining representative samples;

j. A description of the sampling, handling, and preparation techniques, including techniques used for extraction, containerization and preservation of the samples; and

k. A description of the tests performed including results.

Id.

4. Agency Determination

In considering whether such a variance will be granted the agency will look at the design and operation of the treatment system being used. The agency will then determine if the technology and operation are consistent with BDAT. *Id*. The Administrator will publish public notice in the Federal Register of EPA's intent to approve or deny the petition and provide an opportunity for public comment. 51 Fed. Reg. 40606 (1986). *Id*. at 40642 (to be codified at 40 C.F.R § 268.44).

VI. REGULATORY LIMITATIONS ON TREATMENT OR RECLAMATION

A. General

EAF dust may well be a source of reclaimable metals including zinc and lead. Processes designed to accomplish a separation of the valuable metals contained in EAF dust are under development. This research and development effort is driven in large part by the forthcoming land ban and by the desires of companies to eliminate or mitigate long term expenses and risk associated with land disposal. However, both on-site and off-site treatment and reclamation carry with them their own baggage of potential liabilities and regulatory requirements.

B. Status Of Dust As It Relates To Reclamation

As indicated above, EAF dust is a listed hazardous waste classified as a sludge. Thus, EAF dust remains subject to regulation as a hazardous waste under RCRA during the transportation and storage prior to reclamation. Further, generators remain subject to generator requirements even though their EAF dust is destined for reclamation. 40 C.F.R. §§ 261.2(c)(3) and 261.6(b), (c) (1985). Further, generators cannot offer the EAF dust to transporters or to a storage facility which does not have an EPA I.D. number. *Id*. § 262.12.

C. Product of Reclamation

The various treatment processes under development are designed to reclaim the metals from EAF dust. These processes may vary significantly as to the quality of the product. If the metals removed are suitable for direct use or must only be refined, the products of the reclamation process are not waste. If, however, the material is only minimally reclaimed or the product requires further reclamation to extract the metals in a usable form, the intermediate product remains subject to regulation as a solid waste. 50 Fed. Reg. 634 (1985); 40 C.F.R. § 261.3(c)(2) (1985). Thus, even the storage and transportation of the intermediate product may remain subject to regulation.

EPA has established a procedure whereby the owner/operator of the treatment or reclamation process can pursue a variance from regulation of the product as a solid waste. The variance should be available in those cases where the partially reclaimed material is commodity-like notwithstanding the need for additional processing. 40 C.F.R. § 260.31(c) 50 Fed. Reg. 655 (1985). The criteria which will be employed by EPA in considering applications for variances for such partially reclaimed materials are as follows:

1. The degree of processing that the material has undergone and the degree of further processing required;

2. The value of the material after it has been reclaimed;

3. The degree to which the initial reclaimed material is like an analogous raw material;

4. The extent to which an end market for the reclaimed material is guaranteed;

5. The extent to which the reclaimed material is handled to minimize loss; and

6. Other relevant factors.

40 C.F.R. § 260.31(c) (1985).

D. Byproduct or Waste Residue from Reclamation of Treatment

Commonly, a waste residue or byproduct is produced from the treatment or reclamation process. The regulatory status of such material may be defined in part by its destiny: disposal or reuse. If the material is to be disposed of, the "derived from rule" comes into play. 40 C.F.R. § 261.3(c). As described above, the "derived from rule" begins with a presumption that any solid waste generated from the treatment, storage, or disposal of a hazardous waste is a hazardous waste. In the case of a hazardous waste that is hazardous merely by virtue of a characteristic identified in Subpart C, this presumption is rebuttable. If the solid waste derived from the hazardous waste no longer exhibits the characteristic of a hazardous waste, it is no longer regulated as such. In contrast, a waste derived from a listed hazardous waste, such as EAF dust, retains the stigma of the listing. The solid waste derived from EAF dust must be formally excluded under the delisting process set forth in 40 C.F.R. §§ 260.20 and 260.22.

If the byproduct from the process is to be reused without further reclamation, it is not a solid waste at all and therefore not a hazardous waste. Id. § 261.3(c). This exclusion however does not apply in the case where the reuse is a practice constituting disposal. Such activities would include the application or placement of the material on land in a manner where the wastes or its constituents might enter the environment. Obviously, the use of the byproduct or residue as fill does not exclude it from regulation as a hazardous waste.

E. Issues Related To The Evaluation Of Treatment Techniques

As steel makers investigate the various processes for treatment and reclamation of EAF dust, a number of issues arise. How can the company limit liability for the improper handling of materials at the treatment or reclamation facility? How can the companies insure that the entity in the best position to insure against risk is responsible for the risk? Most of the issues confronting the companies relate to environmental liability. Of course, as discussed above, the generator must insure compliance with the generator requirements even if the EAF dust generated is destined for reclamation. Further, the generator cannot consign a shipment of EAF dust to a transporter or destination which does not have an EPA identification number. Thus, the generator must take responsibility for compliance with RCRA and compliance is the best means of limiting this type of liability. What about other types of liability?

F. Sources of Generator Liability

The most significant risk associated with shipping hazardous waste to an off-site location which is neither owned nor operated by the generator is that the facility may fail to provide for the proper storage, handling, treatment or disposal of the material. As a result, the site might become the subject of an action to force cleanup or recover cleanup costs brought by EPA, a state or private party. There are several sources of liability which can be imposed upon a generator under federal law in such cases. Further, the presence of a waste at an improperly managed facility may be alleged to be the basis of tort liability for personal injury to neighbors.

Section 7003 of RCRA provides that EPA may bring suit against a generator who has contributed to the handling, storage, treatment, transportation or disposal of a solid or hazardous waste which presents an imminent and substantial endangerment to human health and the environment. EPA can seek an order requiring that such person take such action as may be necessary to eliminate the endangerment. 42 U.S.C.A. § 6973 (Supp. 1985).

Broader authority exists under the Comprehensive Environmental Response Compensation and Liability Act as Amended by the Superfund Amendments and Reauthorization Act of 1986. Comprehensive Environmental Response, Compensation and Liability Act of 1980, Pub. L. No. 96-510, 94 Stat. 2767, 42 U.S.C. § 9601 et seq. (1982), as amended by Superfund Amendments and Reauthorization Act of 1986, Conf. Sub. H.R. 2005, 99th Cong., 2d Sess., 132 Cong. Rec. H9032 (Oct. 3, 1986) (hereinafter cited as "CERCLA"). Section 107 of CERCLA identifies a generator of waste as a responsible party liable for the cost of removal or remedial action necessary to eliminate a condition which may pose an imminent and substantial danger to public health or the environment at a facility to which it sent waste for disposal or treatment. 42 U.S.C.A. § 9607 (1983). In both cases, liability can be imposed upon a generator notwithstanding the fact that the condition which must be remedied was caused solely by the owner/operator of the facility and without regard to the amount of material actually sent to the facility. Similar sources of liability exist under state law throughout the country. For example, section 8 of the Texas Solid Waste Disposal Act contains a provision analogous to section 107 of CERCLA authorizing the Texas Water Commission to order a generator to eliminate a condition which presents an imminent and substantial endangerment to the public health, welfare or the environment. TEX.REV.CIV.STAT.ANN. art. 4477-7 §8(g) (Vernons Supp. 1986).

G. Alternative Arrangements for Mitigating Liability

1. General

While a generator cannot contractually eliminate the liability which exists under state and federal law, the generator can act to shift the financial impact of liability by contract. Obviously, in considering the question of who is in the best position to protect against conditions which may give rise to such liability, the answer must be the owner/operator of the treatment or reclamation facility. Thus, primary liability should be shifted to that entity by contract. This goal can be accomplished through representations and indemnification. The generator will retain such residual liability as may exist depending upon the financial strength of the owner/operator.

2. Compliance and Indemnity

Any test of treatment or reclamation technology and any long term processing arrangement should be documented by contract. The contract should impose upon the owner/operator the obligation to comply with all local, state, and federal environmental statutes and regulations and to indemnify the generator of the waste from any liability resulting from the owner/operator's failure to so comply. Further, the owner/operator of the facility should indemnify the generators from any liability arising from the release of any hazardous substance, pollutant or contaminant from the facility whether or not such release gives rise to a violation.

3. Inspection

In addition to the requirement of compliance and indemnification, the generator should reserve under the contract the right to inspect the facility upon notice. Given the fact that the generator may well have residual liability notwithstanding the indemnification, a periodic inspection of the facility should be conducted. If it is apparent from the inspection that the facility is not in compliance with applicable laws or is not taking action necessary to control releases, the generator would be well-advised to remove all unprocessed material from the facility immediately.

4. Certification of Treatment

The contract should establish a procedure whereby the facility certifies that material delivered to it is completely processed. This requirement is analogous to the certification of destruction commonly issued in the incineration industry. Such certification may well be an important piece of evidence in an attempt to limit the liability of a generator if the facility should become a "Superfund" site.

5. Segregated Storage and Testing

Another device which should be considered is the performance of discrete or separate tests limited to one company's waste. In this way the generator can document that its waste is completely processed and not commingled with the waste of other companies. This device may be useful in the evaluative testing for various treatment technologies but would probably be unduly burdensome and unacceptable to a fullscale processing facility in commercial operation.

6. Transfer of Title

The generator should consider the pros and cons of actually transferring title to the EAF dust to the processing facility. If the function of the processing facility is to remove valuable metals from the dust through reclamation, the sale of EAF dust to the facility may operate to limit downstream liability for the handling of the product which, dependent upon its purity and use, may still remain a hazardous waste and for the disposal of residue from the process.

VII. CONCLUSION

While there may well be a technical debate as to the degree of hazard posed by the handling and disposal of EAF dust, EPA continues to subject this waste to the full regimen of hazardous waste controls. Further, the ban on land disposal of certain hazardous wastes will become applicable to the EAF dust in 1988. This technology forcing prohibition is creating an incentive for the industry, long a leader in recycling, to look for ways to treat or otherwise reclaim or recycle this large volume waste stream. Even these processes can be subject to the cradle-to-grave regulatory approach which pervades hazardous waste management. Active involvement in treatment and recycling demonstrations and establishment of open lines of communication with EPA concerning the results of such studies are important steps in lessening the regulatory burden on the industry. As apparent from the scope of the issues reviewed herein, the decisions confronting the industry are numerous and complex. Further, they are unavoidable. Hopefully, this analysis of the issues will provide a basis for that decisionmaking.

P.R. #5006.6
File #5006.2

APPENDIX II

"Recycling Furnace Dust"

by R. Kaltenhauser

p 31 - 34, "Disposal, Rcycling and Recovery of Electric Furnace Exhaust Dust", Iron and Steel Inc., 1987.

RECYCLING FURNACE DUST

R. Kaltenhauser

A Review of the PTD Sessions at the 44th Electric Furnace Conference

Approximately 200 people attended three sessions to hear 14 technical papers[1-14] presented on the subject of recycling furnace dust. The sessions were organized by the Process Technology Division of the Iron and Steel Society. This article will give a brief review of the information presented by the authors, along with some additional broader conclusions by the editor. Since the subject is very timely, it is desirable to disseminate this information prior to the publication of the complete papers in the Electric Furnace Conference Proceedings.

SUMMARY

The technical status and operating experience of 12 processes for the recycling and/or recovery of furnace dust were reviewed. Some of the processes have a considerable production history, either with furnace dust or with similar materials. The requirements of a particular steelmaker are usually site specific; that is, the process requirements may be determined by the physical location of the dust-producing facility. Recent regulations by the United States Environmental Protection Agency (EPA) require major changes in the procedures for the storage, transportation and treatment of furnace dust.

The Center for Metals Production (CMP) recently published an in-depth report on furnace dust.[17] More recently CMP has organized a group of more than 20 steel producers to sponsor an additional production-scale dust processing test with the two processes considered to be the most viable. The St. Joe Flame Reactor[8] and the Tetronics Plasma Furnace[14] were selected from the 13 processes studied. The first process concentrates Pb and Zn oxides into a product suitable for use in other St. Joe operations, and makes an iron-bearing slag that is suitable for cement aggregate. The second process concentrates Pb and Zn oxides, recovers liquid iron and produces a nonhazardous slag. When stainless steel furnace dust is charged, liquid Cr, Ni and Mo are also recovered.

BACKGROUND

Steelmaking electric arc furnaces generate approximately 2 percent of their charge materials as flue dust, which is usually captured in a baghouse. The dust is typically a mixture of very small particles (10 micrometers) of various metal oxides. The dust composition changes with the different charge materials to the electric furnace. For example, when galvanized scrap is charged, the dust contains higher amounts of Zn. Since stainless steel shops produce dust with significant amounts of Cr and Ni, it is desirable to process the dust to recover these metals. Although iron oxide is the major dust constituent, the content of heavy metals such as Pb, Cd and Cr is of more significance, since these metals are considered toxic by the EPA. Dust processing must now be selected to meet the EPA regulations rather than only for economic considerations. Some of the considerations for dust processing and the problems with some of the recycling processes were reported in 1985.[18]

An excellent background paper presented during the Conference describes the necessity for the development of environmental controls for industrial wastes.[1] Since conditions of dust generation and requirements for dust disposal vary considerably in different steel plants, the selection of a recycling process will probably be based on individual site requirements. The specific language of the current EPA regulations dealing with electric furnace dust is clearly presented in another paper.[3] A large quantity of dust is generated each year (approximately 500,000 tons) and the dust contains toxic metals which may be transported by ground water into public drinking water supplies. For this reason the EPA has promulgated a series of regulations to:

- Define the responsibilities of the generating steel plants
- Regulate the storage, transportation and treatment of electric furnace dust
- Control the final disposal or use of the dust

Since the primary current disposal method is storage in controlled landfill sites, the EPA date of August 8, 1988 is highly significant. After that date, landfilling of electric furnace dust will not be permitted. The EPA also must define the "best demonstrated available technology" for the treatment of electric furnace dust. Specific EPA definitions deal with the recycling, mixing, treatment, by-products derived from and final disposal of electric furnace dust.[5] Steel producers must become familiar with these regulations and their own liabilities as the generators of the dust, and must carefully make decisions to provide for the lowest-cost disposal of the dust in the future.

AVAILABLE PROCESSES FOR DUST TREATMENT

The processes discussed in the technical papers can be subdivided into several groups for ease of comparison. Some overlapping among the groups will be apparent, although I have attempted to highlight the most viable features of each process.

Recycling Processes[2,3]

Integrated steel producers have long been adding furnace dusts to their sinter mixture to prepare blast furnace charge material. Unfortunately, the fraction of furnace dust in the sinter is so small that no quantitative data has been obtained on the recovery of metals from the furnace dust. If a sinter plant is not available to consume furnace dust, another method of consolidating the dust may be used.[2,4,7,11] The blending of dust with other mill wastes, such as mill scale or grinding swarf, generally increases the strength of the consolidated form. Dust from AOD convertors may contain high levels of lime (10 percent) and require slaking of the lime prior to attempting consolidation. A range of binders has been reported, such as water, various organic compounds or silicas. Curing conditions may range from air to drying ovens to steam. The subsequent material handling conditions in various plants may require different strength levels in the pellets or briquettes.

Mass balance calculations on the charge materials must be carefully made to determine what happens to recycled dust. If the metal oxides are to be reduced, a reductant, reaction time and reaction energy are required. Carbon may have been added to the pellet or will be taken from the liquid melt. The reaction kinetics will be influenced by such factors as the location of the carbon, pellet size, pellet density and similar basic considerations. Approximately three times more electrical energy is required to melt and reduce metal oxides compared to only melting metal. The percent of recovery of metals from the dust will be determined from the combination of such factors. The unreduced metal oxides will go into the newly generated furnace dust, the slag and the metal. It is typical to experience an increased rate of dust generation when consolidated dust is a charge material. Several producers have experienced similar problems to those reported at this meeting.[14] It is difficult to collect accurate data on the quantities of dust produced, electrical energy used and similar important data which is necessary to make mass balance calculations. Such data may have to be collected over an extended period of time to identify exactly what is happening during recycling. Unfortunately, some of the data in these papers was based upon limited testing.

The most important question regarding recycling is where do the Zn, Pb and Cd go? As recycling is continued, the levels of these volatile metals will increase in the dust. At some point the higher levels in the charge may affect the metal bath composition. This could result in excluding recycled dust from the furnace charge when making some grades of steel. This, in turn, requires charging increased amounts of recycled dust into the remaining grades of steel. Will the shop environment remain stable? An alternative is to market the dust when its Zn content has increased, but before the level of other impurities has increased sufficiently to decrease the value of the dust to the Zn producer. This level would have to be negotiated with the Zn producer. Will it change as new EPA regulations are enacted or as the economics in the zinc industry change?

Recovery of Pb and Zn[4,6,7,9]

Some of these processes have been in commercial operation for many years and all of the processes are in operation in production-size installations. Each of the processes requires multiple operations. Economy of scale is also a consideration. Some minimum-size operation is required to achieve a favorable cost. These plants may not be economical for on-site operations in small dust generating steel plants. However, area dust recovery centers to serve several plants are in operation.

The Waelz Kiln process has been in commercial operation since 1923 to concentrate zinc from low grade ores. One plant in Germany has been operating on 100 percent electric arc furnace dust since 1977. Similar facilities are in operation and under construction by HRD in the U.S.[9] and 150,000 tons of electric arc furnace dust were processed in 1986. The dust is mixed with a reductant, such as coke breeze and limestone, and is fed into a rotary kiln. Reduction of the metal oxides occurs, and the volatile metals are reoxidized in the kiln atmosphere and are collected at a typical concentration of 55 percent Zn. The slag successfully meets the EPA leach test requirements and can be used as road fill. The recovery of iron is not economical with the present process.

The Michigan Tech cold bonding pelletizing process is the basis for the PTC carbon-bearing process.[4] There has been extensive industrial experience using a blend of various steel plant waste metal oxides. The oxides are pelletized with carbon fines, lime and silica; hardened in an autoclave; metalized in a rotary kiln; and melted in a cupola or blast furnace. During metalization, oxides of Zn and Pb are collected from the exhaust. When using 100 percent electric arc furnace dust pellets, Zn was concentrated from the initial 19 percent in the feed to 64 percent in the fume. The metalized pellets can then be melted to recover liquid iron and slag. Tests have also been made using stainless steel dust to recover Cr, Ni, Mo and Fe. An 80,000-ton-per-year demonstration plant and a 1.5-ton-per-hour pilot plant are being constructed in upper Michigan.

The INMETCO process[7] has been in operation since 1978 in Ellwood City, PA, using stainless electric arc furnace/AOD dust (18,000-tons-per-year) blended with other steelmaking metal oxide wastes. The sized wastes are mixed with coal fines for reduction and pelletized. The greenball strength is sufficient for direct transfer into the hearth of a rotary

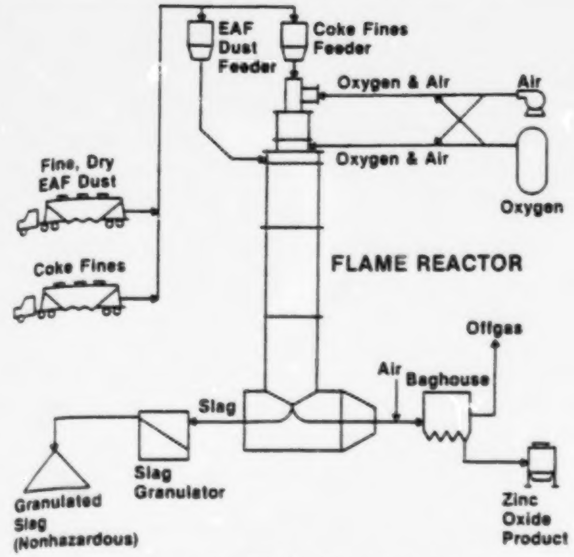

Fig. 1 — The St. Joe Flame Reactor.

furnace. During metalization, oxides of Zn and Pb are collected in the exhaust. The metalized product is transferred hot into a submerged arc melting furnace for the recovery of Ni, Cr, Mo and Fe. The nonhazardous slag can be used for road fill.

The St. Joe Flame Reactor[8] has been in operation since 1983 in Monaca, PA, using a variety of metal-bearing waste materials as feed to a 20,000-ton-per-year demonstration plant. Fine coal is injected into a water-cooled burner operating with oxygen-enriched air (Figure 1). Dry metal oxides are injected into this reducing, high-temperature flame. The volatile metals are reoxidized and collected in a baghouse. Iron is not reduced but is removed in the slag; this is suitable for cement aggregate since it passes the EPA leach test. Operation with more than 170 tons of electric arc furnace dust concentrated the 12 percent Zn in the feed to a level of 40 percent in the baghouse oxides.

St. Joe has engineered and planned an on-site 20,000-ton-per-year electric arc furnace dust facility, which can be built and operated by its Minerals and Resources Recovery Corporation. Waste management services such as this could relieve the steel producer of the responsibility for disposal of his electric arc furnace dust without creating possible future liability. Details are also given for the capital and operating costs of such facilities. (At 2 percent dust generation, a 20,000-ton-per-year plant could process the dust from a 1 million-ton-per-year electric arc furnace steelmaking operation.)

Hydrometallurgical Processes[9,10]

A process for the treatment of combined electric arc furnace dust and spent acid wastes is in operation at Atlas Steel.[9] The objective of the process is to dispose of both waste dust and spent acids. The dust is first acidified with the iron-bearing spent acids; lime is added to form precipitates; and then slag is added to solidify the remaining liquid as complex silicates. Since the raw materials must be combined in specific proportions, the plant wastes will have to be supplemented with additional purchased materials. Leach tests on the solid product were reported to meet the EPA required levels, but the process has only been in use in Canada and more recently in France. The author prudently points out the difficulties in obtaining a delisting of a treated product by the EPA. An economic evaluation of capital and operating costs for a 12,000-ton-per-year operation is also given.

A caustic soda leaching/electrowinning process[10] to recover Zn from electric arc furnace dust has been operating in France since March 1986. The plant is designed to process 13,200 tons per year of dust. The dust is mixed with used electrolyte and separated into magnetic and nonmagnetic components for separate leaching treatments. Solids are centrifuged, water-soluble constituents are washed out and the solid residue is wasted. Precipitation of silicates is achieved with an addition of lime. Lead and copper are then recovered by cementation with Zn powder. The powdered Pb is marketable. The remaining caustic solution is then electrolyzed to recover the Zn.

The various rinse waters and salts obtained during the process are mixed with spent pickling acid from an adjacent galvanizing plant. No EPA-type information was given for the solids produced and the value of the waste acids was not considered in the economic data. It appears that a specific market for the electrolytic Zn powder exists in France; however, the market value for other applications is not given.

Reduction Processes to Recover Metals and ZnO[11-14]

The Scandust process[11] has been operating in Sweden since 1984 at a 77,000-ton-per-year plant for steelmaking metal oxide wastes. Electric arc furnace dust plus other metal oxides are mixed with coal fines and fluxes, and are injected into a plasma flame in a coke-filled shaft furnace. Three 6 MW plasma torches are located in the tuyere area of the shaft furnace. The volatilized Zn and Pb are collected in spray condensers which separate the two metals. Liquid Fe and nonhazardous slag are tapped from the furnace hearth. Cr, Ni and Mo were also recovered when stainless steel dusts were processed for the first two years before the plant changed to carbon steel dust. Materials consumed, yields and the operating availability of the equipment are given. An important part of the net operating cost is the sale of by-product energy to an adjacent city government.

The relatively large size of this facility requires that the dust and other charge waste materials be transported to it. The regulations and costs for this and the domestic cost for electric power ($0.03/kWh) are specific to the Swedish location. In a meeting subsequent to this Conference, plans were discussed for a smaller 44,000-ton-per-year facility in the United States. A similar-size plant is also in operation in Sweden and it is producing ferrochrome from chromite ore. As a result of the relatively long production experience, capital and operating cost information should be reliable.

The Tetronics Plasma Furnace[14] has been in commercial operation for several years in locations around the world. Two ferrochrome plants, one titanium sponge plant, a platinum recovery plant and a nickel recovery plant are in operation. The research furnace and a 6,000-ton-per-year pilot furnace are located in the U.K. (Figure 2). Both carbon and stainless steel furnace dust have been processed in sizable quantities. A water-cooled 2 MW plasma torch in the furnace roof rotates to keep the bath liquid. The waste material, mixed with coal and flux, is screw-fed through the roof and falls into the liquid metal-slag bath. Strongly reducing conditions in the furnace and the well-distributed plasma flame result in the separation and recovery of the volatile Zn and Pb, the recovery of liquid iron (plus Cr, Ni and Mo from stainless dust) and a nonhazardous slag. The considerable operating experience with six furnaces and a variety of metal oxides has resulted in reliable capital, maintenance and operating costs data. A furnace with a single 2 MW plasma torch can be operated to process up to about 8,000-tons-per-year of waste metal oxides. This size unit should be economical as an on-site facility for individual electric furnace steel plants. For smaller producers, the additional furnace capacity could be used for other plant wastes.

The Plasma Extended Arc Reactor process[12] has been scaled up from a laboratory unit at the University of Toronto to a pilot plant facility powered by a 1 MW plasma torch which can reduce furnace dusts at a feed rate of 1 ton per hour. The extended flame is produced by the plasma gas injected through the center of the hollow graphite electrode. The dust is agglomerated with coal and fluxing agents, fed into the top of the vessel and allowed to fall through the plasma zone. The vaporized Pb and Zn are recovered in a Zn splash condenser. Liquid iron and slag are tapped from the furnace hearth. Trials have also been made using flue dust from stainless producers with recovery of Cr, Ni, Mo and Fe. Estimates of capital and operating costs have also been made.

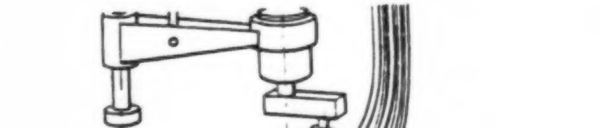

Fig. 2 – The Tetronics Plasma Furnace.

Although not included in these technical sessions, laboratory, pilot plant and production work with several similar plasma processes have been recently reported. The PEC plasma furnace[15] is operating with iron ore fines in Spain, with various wastes in Canada, and a new facility for furnace dust will begin operation in Italy in early 1987. The Plasmacan developmental furnace[16] has been operating with a number of waste feed materials and a production facility has been designed.

The Flash Direct Radiation process[13] has been placed in this group with the plasma processes, although it does not provide for the separation of metal and slag. The waste oxide feed material, mixed with a volatile reducing agent such as coal or sawdust, is rapidly heated in a hot-tube reactor. The reduced Pb and Zn are quenched into shot when they pass out of the high temperature reaction zone. The remaining metals and slag constituents are also formed into shot. Pilot-scale tests have been conducted in a 12-inch diameter reactor in Spain at a dust feed rate of 5 lbs/minute. Additional testing is required to obtain complete removal of the Pb from the slag-metal shot. No cost estimates were given.

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P.R. #5006.6
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APPENDIX III

"So Little Time, So Many Choices"

by T.P. McAloon

p 14 - 20, "I & SM", August 1987.

So Little Time, So Many Choices

One year from now the land disposal of electric arc furnace dust will be banned by the EPA. Steelmakers are now in the process of deciding which dust disposal option is right for them.

by T.P. McAloon Managing Editor

A lot of people worry when the planets of our solar system line up in a row like so many billiard balls. They believe the combined gravitational forces have the potential to wreak havoc upon the earth. Pretty silly, right? Well, a lot of electric arc furnace operators are worried about the number eight lining up in a row to form 8/8/88. Equally silly, right? Not really. On that date the land disposal of electric furnace dust or sludge in the United States will be banned by the Environmental Protection Agency (EPA) and the gravitational pull of this law is sure to wreak havoc upon cash and process flows alike.

While you have a few years to prepare for the next syzygy as it is called, electric furnace steelmakers have just one year left to decide how they will dispose of their dust. Steelmakers are currently besieged by a number of suppliers promoting a diverse array of dust disposal methods. To help them sort through their options, the Center for Metals Production (CMP) is conducting two demonstration tests and it recently sponsored a meeting in Pittsburgh, PA, to discuss other treatment technologies. Before we look at the processes discussed, however, some background information is required.

HOW DID THIS ALL START ANYWAY?

Congress Told the EPA to Get Tougher

Dust is such a common part of our everyday lives that it becomes invisible to most of us. Yet it is in the air we breathe and the water we drink. Indeed, the interaction that occurs between electric furnace dust and groundwater is the deepest root cause of the dust disposal crisis. Because the toxic constituents of the dust, e.g., lead, chromium and cadmium, are absorbed by groundwater, the EPA had no choice but to list it as a hazardous waste under the stringent new guidelines imposed by Congress.

In 1984 Congress passed the Hazardous and Solid Waste Amendments to the Resource Conservation Recovery Act. Congress expressed in the amendments a very strong sentiment against the further land disposal of untreated hazardous wastes. To ensure that its wishes were quickly carried out, Congress added the so-called "hammer provision," or land ban deadline. The intent of the deadline is to insure that by a certain date, August 8, 1988 in the case of electric furnace dust, land disposal will be banned *unless* proper new regulations are developed prior to that date by the EPA.

The Steel Industry's Lobbying Effort is Launched Soon Afterward

The American Iron and Steel Institute (AISI) and its member companies have been involved in an intensive effort to influence the regulations that will be developed for electric furnace dust disposal. They hope to convince the EPA that land disposal of 500,000 tons of dust per year can be done in such a way as to insure the protection of human health and the environment. Baker/TSA was commissioned by the AISI to prepare a report demonstrating that land disposal of electric furnace dust in a monofill, where only one type of material is stored, can meet the EPA's requirements. The report was based upon data from disposal operations at a number of member companies' facilities.

Following the completion of the report, the AISI submitted a petition to the EPA. It requests that rules based on the Baker/TSA report be established to permit the land disposal of dust in properly-designed facilities. The EPA has yet to comment on the petition.

Earle F. Young Jr., the Institute's vice president — environment, summed up the situation like this. "We have great hopes that the EPA will, in a timely fashion before the dread target date of August 8, 1988, promulgate (our suggested) regulations. If this is done, we believe that adequate facilities can be installed and that land disposal of the waste can continue. However, that's far from a sure thing."

A Second Front is Opened Just in Case

Because of the uncertainty over the EPA's response to the petition, the industry has been attacking the problem on a second front — treatment of the dust. The CMP, which is a research and development applications center established by the Electric Power Research Institute (EPRI), has been overseeing two dust treatment demonstration projects since late 1985. The two technologies are:

(1) The St. Joe Flame Reactor (Figure 1)
(2) The Bethlehem/Tetronics Plasma Process (Figure 2)

Both of these techniques are thermal processes in which the electric furnace dust is reacted with carbon to reduce the oxides of zinc, lead and cadmium (if present). These metals are then volatilized out of the reactor leaving an inert nonhazardous metal-slag mixture. The metals can then be marketed and the metallic slag disposed of by conventional means.

These two approaches were chosen for demonstration by 22 steel companies and one industrial gas supplier (Table I). The participating companies contributed $30,000 each to the project and EPRI matched these funds for a total shared cost of $1.2 million.

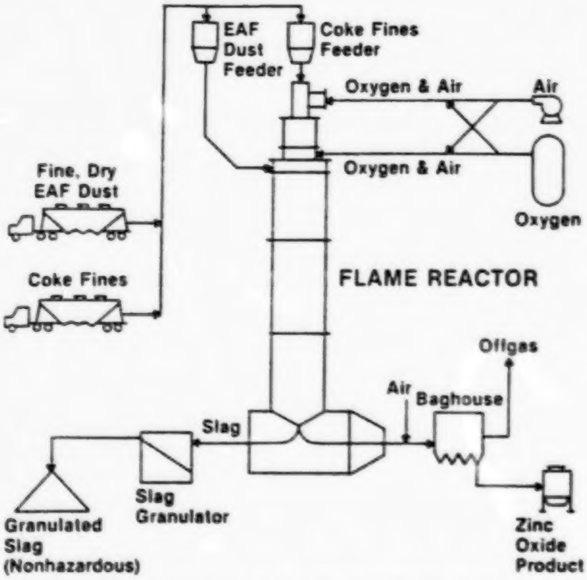

Fig. 1 – Schematic of the St. Joe Flame Reactor.

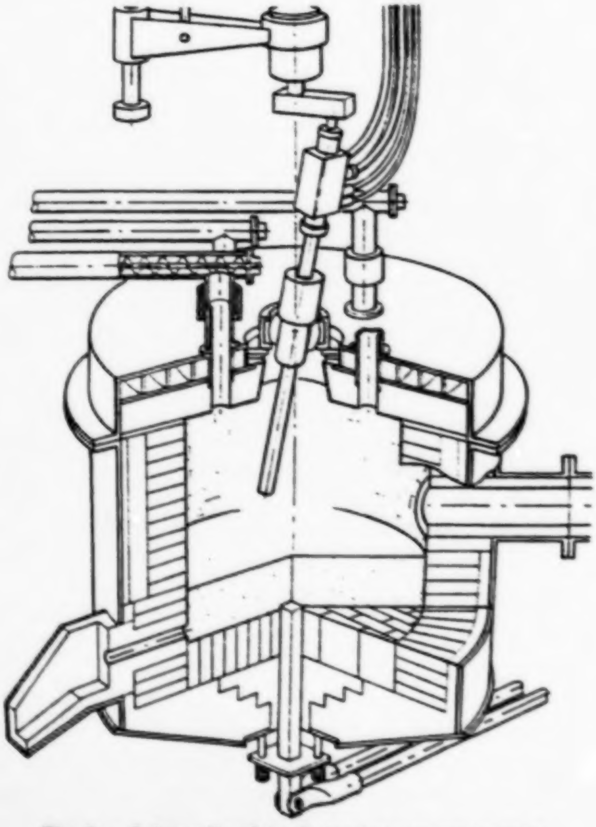

Fig. 2 – Schematic of the Tetronics plasma furnace.

Table I – Participants in the CMP Dust Project

Armco Inc.
Auburn Steel Company
Babcock & Wilcox
Bethlehem Steel
Chaparral Steel
Empire Detroit
Florida Steel
Hoeganaes Corporation
Inland Steel
Ivaco
Linde Div., Union Carbide
LTV Steel Company
Lukens Steel Company
National Steel Company
Northwestern Steel & Wire
North Star Steel
Nucor
Raritan River
Stelco
Structural Metals Inc.
TAMCO
The Timken Company
USX

EPRI alone is paying for all project management and communication costs.

According to Joseph Goodwill, director of the CMP, the results of the test program will be made available to the public in the first half of 1988. For now, however, the preliminary test results are known only by the participating companies.

SO MANY CHOICES, SO LITTLE TIME

Do not think that electric furnace operators have narrowed their dust disposal options down to the two processes being tested. They have not. A number of other processes and approaches are also being considered, as evidenced by the larger-than-anticipated turnout for the CMP's "Update on EAF Dust Activities" meeting. Over 125 people representing approximately 41 steelmakers and numerous interested parties attended the meeting, which was held on June 18, 1987 at the Airport Holiday Inn in Pittsburgh.

Nine approaches to the dust disposal problem were featured at the meeting:

(1) The Elkem Process
(2) TIBUR-Howden's PEAR Process
(3) Horsehead Resource Development's Expansion Plans
(4) SKF's Plasmadust Process
(5) Stainless Steelmaking Dust Treatment by the Tetronics Plasma Process
(6) Dust Injection Trials at LTV Steel
(7) The Nippon Magnetic Dressing System
(8) Davy McKee's Hi-Plas Process
(9) The Chemfix Process

Presentations were limited to commercial plans and economics rather than technical detail. The following sections feature the highlights of each presentation.

Elkem Enters the Fray

Elkem's process for the treatment of electric arc furnace dust (Figure 3) combines two commercially-proven components into one system:

(1) Elkem's air-tight multipurpose electric furnace technology for volatilizing the dangerous metals out of the dust

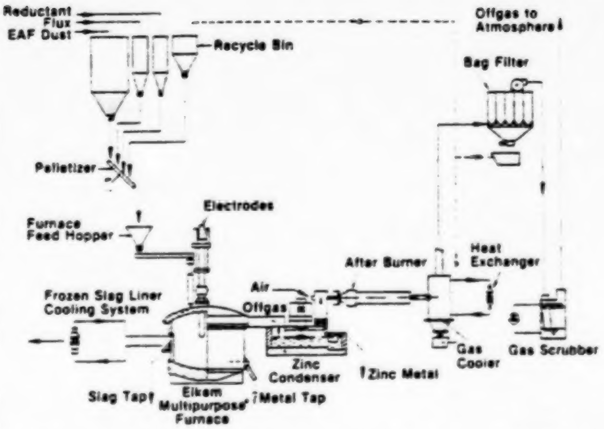

Fig. 3 – Schematic of Elkem's electric arc furnace dust treatment system.

(2) Imperial Smelting Processes, Ltd.'s zinc-condensing technology for recovering zinc metal from the offgas

The objectives of Elkem's process are the production of a delisted, nonleachable, disposable slag; to recover zinc and lead metal; and to recycle all solid emissions from the process and neutralize all process emission gases. The system is designed for in-plant use.

Elkem is one of the late entrants into the dust disposal sweepstakes. At present it has built only a small (55 kg of dust per hour capacity) pilot scale demonstration system that is a simplified version of its commercial processing scheme. A three-day demonstration of the process was made recently before Norwegian and American steel producers, representatives of the zinc industry and members of the academic community. A zinc condenser efficiency of about 55 percent was achieved. Despite the disappointing condenser efficiency, Elkem pronounced the demonstration test to be a success. It now hopes to supply a commercial system.

TIBUR-Howden's PEAR Process Presses On

TIBUR-Howden's Plasma Extended Arc Reactor (PEAR) is, as the name suggests, a plasma-based process that uses graphite electrodes to generate a plasma flame. Like Elkem's process, it is designed to recover valuable materials from the dust (zinc and lead) and produce a nonhazardous slag.

The PEAR reactor vessel has a 3- to 4-foot inside diameter, a 4-foot cavity height and has three electrodes projecting inside (Figure 4). The electrodes are spaced 120 degrees apart and they are 15 degrees from the horizontal. The plasma gas is introduced into the vessel through longitudinal holes drilled along the electrode's axis; research in this area was conducted for 15 years by the University of Toronto.

TIBUR-Howden is pursuing a very measured development program. So far it has built a 1 MW pilot plant. It chose 1 MW so that scale up to a typical on-site facility would not be a significant factor. Company officials believe that 1 to 3 MW would be appropriate for most sites.

Test results are very tentative right now according to Bill MacDonald, Howden Group Canada Limited's project manager of new business development, and he declined to release the company's current zinc recovery numbers at the meeting. However, he did make cost estimates. The estimated dust disposal cost for the PEAR process is $120 to $150 per ton of dust and the power consumption is 1.5 MW per hour per ton of dust. Mr. MacDonald stressed that these numbers are very site specific.

The next step in TIBUR-Howden's development program is the construction of a demonstration unit at an appropriate site. Mr. MacDonald believes that the company will have an on-site demonstration unit operating in six to 12 months. The company wants to supply its technology through an intermediary company; it does not want to be responsible for the day-to-day operation of an on-site PEAR facility.

Horsehead Expands Its Operations

Unlike the previous two companies, Horsehead Resource Development Co., Inc. (HRD) is not selling an on-site treatment facility and is not in the development stages of a process. HRD is the owner/operator of a 250,000-ton-per-year regional facility in Palmerton, PA, that has been treating electric furnace dust since 1980. And more capacity is on the way.

HRD, a subsidiary of Horsehead Industries, Inc., has announced that a second facility capable of processing 75,000 tons of dust annually will be operating by the late third quarter of this year in Calumet City, IL, which is just south of Chicago. The Calumet plant will be capable of receiving dust by barge, rail or truck. A third facility in Enid, OK, is also being investigated to serve the Southwest. A fourth facility at a site to be determined is also being considered.

This year the Palmerton plant alone will process 200,000 tons of dust. Once the new facility in Calumet City is operational, the company says it will have 325,000 tons of capacity within a reasonable transportation cost of most of the electric furnaces in North America.

Under HRD's regional treatment facility concept, all the dust generator pays is a per ton "tipping" or dumping fee and transportation costs. HRD assumes responsibility for treating the dust and disposing of the by-products. The tipping fee depends upon the zinc content of the dust. For dust with a zinc content of 20 percent or greater, HRD charges $45 per ton. If the dust is under 20 percent in zinc content, it charges an additional $3 per ton for every percentage point under 20 percent.

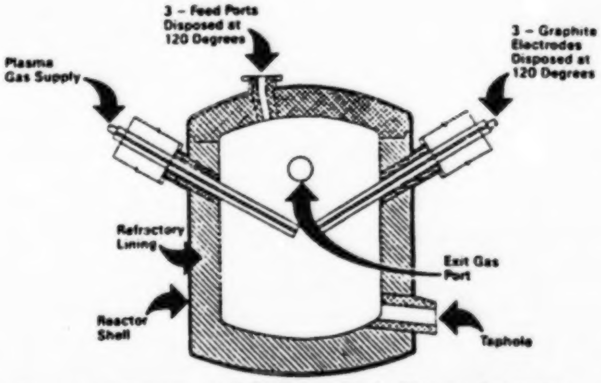
3 – Feed Ports Disposed at 120 Degrees
Plasma Gas Supply
3 – Graphite Electrodes Disposed at 120 Degrees
Refractory Lining
Exit Gas Port
Reactor Shell
Taphole

Fig. 4 – Schematic of TIBUR-Howden's Plasma Extended Arc Reactor.

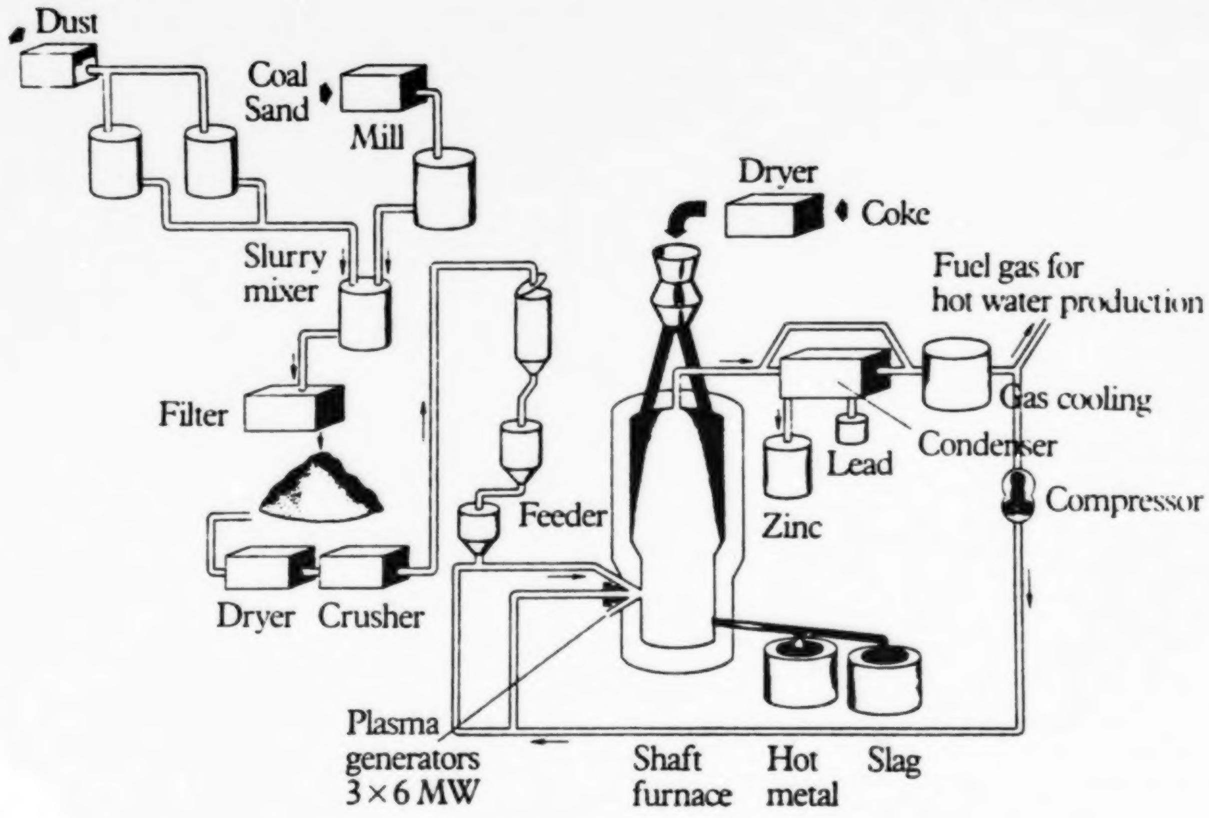

Fig. 5 – Flowsheet of the Plasmadust process.

The Waelz kiln process is used to convert the electric furnace dust into crude zinc oxide and a nonhazardous slag. The zinc oxide produced is used by New Jersey Zinc, another subsidiary of Horsehead Industries, and the slag products are marketed for highway and construction uses. While HRD is the only practitioner of the Waelz kiln process for electric furnace dust treatment in North America, the technology is being used for such purposes in Germany, Spain, Italy and Japan.

The EPA is presently evaluating HRD's Palmerton facility for Best Demonstrated Available Technology (BDAT) status. In fact, HRD's Waelz kiln process is the only process presently under consideration for BDAT status by the EPA.

"The EPA is clearly expressing confidence in our (HRD's) ability to process hazardous waste safely and effectively within environmental regulations," said Clayton Kennedy, a sales/marketing specialist at HRD.

A Plasmadust Facility in the U.S.A.

The highlight of SKF Plasma Technologies' presentation was the announcement that Wheelabrator Technologies, Inc. is planning to build a 100,000-ton-per-year Plasmadust regional processing facility in Canton, OH. The proposed site is very near The Timken Company's Faircrest plant. The total investment cost is expected to approach $60 million and a startup date of late 1989, early 1990 is expected. Startup is contingent, however, upon obtaining the necessary operating permits from the EPA.

The contract between SKF and Wheelabrator should be finalized by the end of this year. Wheelabrator will make its decision to go ahead with the construction in early 1988.

Although the financing arrangements for the project have not yet been worked out, John Caine of Wheelabrator Air Pollution Control described the deal as "quite firm."

It was estimated that the proposed plant's tipping fee would be in the range of $60 to $80 per ton, transportation not included. As with Horsehead's tipping fee, the zinc content of the dust will affect the amount charged.

"If you had as little as 15 percent (zinc content) and you were close by, it might still make sense (to send your dust to the proposed plant). If you had 15 percent and you were 500 miles away, then it's a different problem," said Mr. Caine.

A 77,000-ton commercial Plasmadust plant has been operating at Landskrona, Sweden, since April 1984. Called ScanDust AB, the company is jointly owned by a local investment company, SKF and Uddeholm, a steel company. The remaining interest is held by a private businessman. The Plasmadust Process (Figure 5) uses high-temperature plasma torches to reduce oxidic waste in a coke-filled shaft. The zinc and lead in the dust are vaporized and leave the furnace with the offgas. They are collected as liquid metals in a condenser outside of the furnace. The iron and slag produced by the process collects in the bottom of the shaft, which is tapped very much like a blast furnace.

The process produces zinc of Prime Western Grade as well as iron alloys. The slag produced is used as filling material for construction projects. The Plasmadust Process also produces a great deal of excess energy. At Landskrona the excess gas is used to heat a hot water boiler, which is connected to the local municipality's district heating system. At the Canton site, waste heat will be sent to an industrial-size boiler approximately one mile away.

While the Landskrona plant processes both stainless and carbon steel dust, the Canton facility will not process alloy dust, at least initially.

"We think there are some advantages to focusing and dedicating a plant to just the one type of dust," said Peter Baumann of SKF.

What Options are Available to Stainless Steelmakers?

Stainless steel producers are faced with the same dust disposal problems as carbon steel producers. However, the process requirements and economics of stainless dust processing are somewhat different than those for carbon dust processing. They are different because the cost of treating stainless dust may be substantially offset by the recovery of valuable metals such as chromium, nickel and molybdenum. These metals can be recycled to the stainless steelmaking process.

To obtain data on the process requirements and economics of stainless dust processing, the CMP and Allegheny Ludlum Corporation (ALC) sponsored a study. Robert Kaltenhauser, manager of process development at ALC, was the project's principal investigator.

According to Mr. Kaltenhauser, the study had three objectives:

(1) To briefly review the current processes capable of processing stainless steelmaking dust
(2) To conduct pilot studies on a small plasma system at Tetronics in Faringdon, England, with four 25-pound samples of electric furnace and AOD dust from ALC
(3) To use data from the pilot studies and other large-scale trials (45 tons) run by British Steel Corporation to determine the process economics for installing a 6,500-ton-per-year plasma system in the United States

The results of the study were very encouraging. Mr. Kaltenhauser reported that the operating cost (including capital) of the Tetronics plasma system was $180 per ton of dust for a 6,500-ton-per-year system. This is not the cost to the operator, however. The $180-per-ton charge is more than offset by a projected $138-per-ton gain from the value of the metals recovered and a $160-per-ton savings in landfill expenses. The net savings of $118 per ton represents, said Mr. Kaltenhauser, an after-tax return of 19 percent on the $2 million capital investment.

It should be noted that this projected return on investment (ROI) for the installation is low given the inherent risk associated with implementing a relatively new technology. However, the ROI could improve substantially if current dust disposal costs increase or if it can be demonstrated that the plasma unit is capable of treating other steel plant wastes such as pickling liquor. Finally, the assumptions used in performing the ROI calculation (e.g., tax rate, depreciation schedule, discounting methods, etc.) are the standard method used by ALC, said Mr. Kaltenhauser. Other companies that use a different financial analysis method could get different results.

In the conclusion of the report, Mr. Kaltenhauser recommended that the next step be a site-specific economic study of a Tetronics' installation at a U.S. specialty producer's plant.

The full report on the investigation is titled "Plasma Process Treatment of Stainless Steelmaking Dusts," (CMP report number 87-1) and it can be purchased from the CMP.

The Results of the Dust Injection Trials at LTV

Dr. Peter Koros, senior research consultant at LTV Steel Company's research center, had the distinction of being the only speaker at the meeting who was not selling a process. He was there to talk about what happens when electric arc furnace dust is injected back into the furnace; the trials were carried out at LTV's Cleveland Works.

LTV has been experimenting with various dust disposal options since November 1985, when an LTV-controlled landfill serving the Cleveland and the now-idle Pittsburgh works closed. Currently, most of the Cleveland Works' dust is being sent off-site for briquetting. The remainder is disposed of in a landfill. The briquettes are added in bulk to the plant's two electric furnaces. This results in an increase in the zinc content of the dust.

In 1985, prior to the recycling of the dust and the appearance of large quantities of zinc-coated scrap, the production of 1 ton of steel generated 17 pounds of dust. The average zinc content of the dust was 27 percent. By early 1987, following LTV's decision to add 13 pounds of briquettes per ton, the dust generation rate had climbed to 23 pounds per ton. The average zinc content of this dust ranged from 33 percent to 40 percent, with the average being 35 percent.

The bulk addition of briquettes has a large drawback, however. A great deal of energy is lost when the roof is lifted off to allow the addition. To avoid this problem, it was decided to give dust injection a look because injection systems were already in place on Cleveland's furnaces (Figure 6). The systems had been installed to facilitate the production of low-sulfur and low-phosphorus steel through the addition of lime-spar powder.

According to Dr. Koros, injecting dust does have a process cost; electrical energy is being used to smelt, which is very uneconomical. He reported that

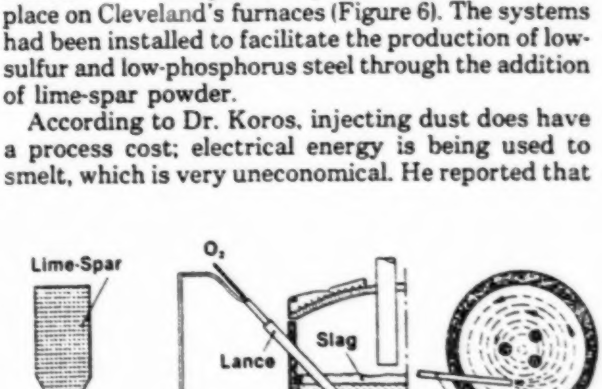

Fig. 6 – Schematic of the LTV Cleveland Works' furnace injection system.

the real power cost of dust injection is 15 kWh per ton. Offsetting this, however, is the possible sale of the zinc-enriched dust to the zinc industry. Indeed, this was one of the driving forces behind LTV's decision to proceed with the dust injection trials.

The effect of dust injection upon the operation of a furnace is not inconsequential, said Dr. Koros. To find out what was happening, some tests were run. First, the furnace slag was sampled to make sure that it was not adversely affected by the dust. It was not. Then the sulfur performance of the furnace was checked. The change was found to be negligible. The effect of the oxygen carrier gas was very noticeable, however. A temperature rise resulted and bath stirring increased. The oxygen loading of the system had a volcanic affect on the furnace, according to Dr. Koros. It seems that the injection process is limited not so much by the dust injection rate but by the ability of the shop evacuation system to withdraw the gases produced. To suppress some of the effects of oxygen and to reduce them to manageable proportions, argon was substituted for oxygen as the carrier gas.

LTV is now planning to put all of this process knowledge to use. It is looking very seriously at injecting 70 percent of the Cleveland Works' dust back into its furnaces. According to Dr. Koros, this would allow the zinc content of the dust to settle around 50 percent to 60 percent, which makes it equal to or better than the feed material now used by the zinc industry. The remaining 30 percent would then be steadily drawn off of the system and sold.

For this scheme to work, however, two things have to happen:

(1) LTV must be able to sell the dust.
(2) The zinc industry must be willing to pay for the dust.

The question that has to be answered, therefore, is: Is it cheaper to process the 25 percent to 33 percent zinc-content dust in order to make it attractive to the zinc industry, or is it cheaper to use the injection process to make dust with a zinc content of >50 percent?

As of yet LTV does not have the answer.

"That's why I'm not selling a process," quipped Dr. Koros.

The Nippon Magnetic Dressing Process

The Nippon Magnetic Dressing (NMD) Process uses the sensible heat of molten slag to convert electric arc furnace dust into a nonhazardous slag and a zinc-rich dust.

The process involves the following steps:

- Dust and coal fines are pelletized, dried and a slag modifier called FUKENITE is added. The mixture is then preheated by exhaust gas from the process.
- The mixture is injected into a reaction vessel for reduction. Heat is supplied by an oil burner and the sensible heat of the slag (1,350° to 1,450°C).
- Hazardous heavy metals dissolve into the molten slag and modifier. They therefore become fixed within the granulated slag.
- A zinc-bearing dust (45 percent zinc) is recovered from the baghouse. This dust is reportedly saleable to zinc smelters as a feedstock.
- The slag produced can be granulated and used for construction purposes.

Beginning in April 1988 an NMD demonstration plant will be operated for three months. The results will be available from Sumitomo Corporation of America, which is the U.S. agent for the technology.

Davy McKee's Hi-plas Process will be Built in North America

During Davy McKee's presentation concerning its High Intensity Plasma Arc System (Hi-plas), it was announced that a demonstration plant will be operating in Canada next year. A 1 MW Hi-plas furnace and a lead condenser for metals recovery will be installed on-site at a steel plant to process electric arc furnace dust.

In the Hi-plas furnace (Figure 7) a DC transferred arc is struck between the water-cooled nonconsumable plasma torch and the molten pool of reactants that surrounds the bottom anode, which is located in the furnace's hearth. A mixed agglomerated feed is then tangentially injected with a nonoxidizing gas into a cyclone reactor, which surrounds a substantial portion of the arc length. The radiant and convected heat from the arc melts the feed in the cyclone reactor; the metal is smelted and the zinc evaporates. A metalized slag is tapped from the hearth and the zinc vapor is sent to the lead splash condenser for recovery.

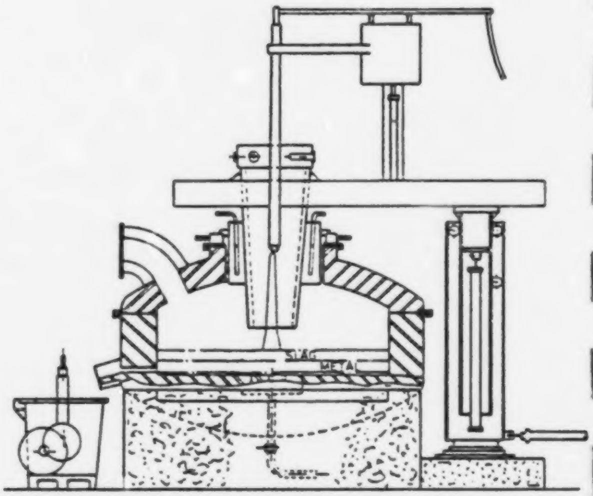

Fig. 7 – Schematic of Davy McKee's Hi-plas furnace.

Davy McKee has been developing its Hi-plas furnace for five years and it has extensive experience in lead splash condenser design as it is a major contractor for the ISP zinc/lead blast furnace process. However, the Hi-plas process for dust treatment will be the first time the two have been put together in one system.

A U.S. Steelmaker Gives Chemfix the Go-ahead

A large American steel company has given Chemfix Technologies, Inc. the money needed to proceed with the delisting of its Chemfix-treated electric arc furnace dust. The go-ahead followed bench-scale work by Chemfix on the dust from two of the company's plants. No dust has been processed by Chemfix on-site, however.

According to Art Buesing of Chemfix, the one-time cost for delisting electric furnace dust should range between $30,000 and $50,000 per plant. It is not known what Chemfix is charging its first dust processing client. Mr. Buesing estimated that a delisting with the EPA should take about one year to complete.

Chemfix is a service company based upon a patented treatment process for the stabilization and chemical fixation of municipal and industrial sludges. It is currently doing work with hazardous sludges at a number of sites.

The process employed by Chemfix uses soluble silicate and silica setting agents like kiln dust, portland cement or fly ash to chemically tie up the heavy metals in the sludge. The product created is nonleachable and resembles a crumbly or friable clay soil. The average volume increase for the treated substance averages 10 percent.

Mr. Buesing estimated that the per ton treatment charge for electric furnace dust would be below $100. Chemfix's processing cost is in the range of $60 to $65; the balance is for solid waste disposal and trucking. Upon the signing of a contract, Chemfix provides its own equipment and people to do the job on-site with a mobile unit (Figure 8) or off-site at a fixed facility.

PROS AND CONS

The nine processes I have reviewed are just the tip of the dust pile. They do give you a good sense of the variety of options available though. You want to send your dust off-site? It can be handled. You want to process your dust on-site? No problem. You want to recover precious metals for possible resale or maybe you don't? You are covered either way.

The presentations made by SKF and Horsehead received the most attention from the audience. It seems there is a strong appeal to just sending your dust somewhere and having someone else worry about it. A possible drawback to this approach is the red tape the EPA requires when you transport hazardous waste. The EPA plans to have a complicated cradle-to-grave manifest system in place. I doubt whether it will be made simpler in the coming years.

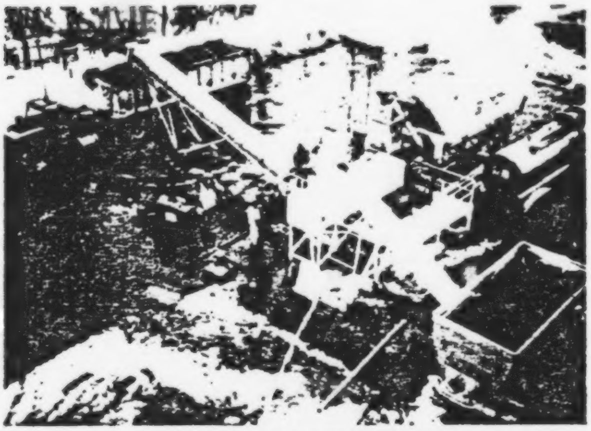

Fig. 8 — Chemfix Technologies' mobile processing equipment.

If you want to avoid the shipping problems and possible generator liability, you can choose to process your dust on-site. Here too the EPA has the red tape out and waiting. The regulations regarding the operation of a treatment, storage and disposal facility are extremely complex. Even the products of your treatment process will have to be checked by the EPA to make sure that they are not hazardous.

Those on-site processes that rely on the sale of recovered zinc to make their operation economical also run another type of risk. The sudden appearance of thousands of pounds of zinc on the market could send prices tumbling. David Naden of Davy McKee even went so far as to predict that the zinc industry will *charge* the steelmakers to take the zinc away! James Jolly, zinc commodity specialist at the U.S. Bureau of Mines in Washington, DC, does not go quite that far, however. He believes that steelmakers will be able to get some value for their zinc from the zinc industry.

"They are not going to make money selling their zinc or zinc oxide but they will get enough to cut their disposal costs somewhat," he said.

Perhaps Joseph Goodwill of the CMP has the best short-term scenario. He believes that a variety of technologies will be applied because of the varying economic conditions in the United States. Whatever the outcome, I am sure of one thing — a lot of lawyers are about to make a lot of money. Maybe I should have gone to law school like my mother wanted.

45th Electric Furnace Conference

December 8-11, 1987
Hyatt Regency Chicago, Chicago, IL

P.R. #5006.6
File #5006.2

APPENDIX IV

List of Contacts

P.R. #5006.6
File #5006.2

APPENDIX IV

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